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THE CHOICE OF

COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.

By A. G. CHARLETON, A.R.S.M.

Excerpt from the Transactions of The Federated Institution of Mining Engineers.

LONDON AND NEWCASTLE-UPON-TYNE:
ANDREW REID & CO., Ltd., PRINTERS AND PUBLISHERS.

1894.

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THE CHOICE OF

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(*a*) That flat pieces of rock cannot pass through it uncrushed to the same extent, as in a machine with reciprocating-jaws.

(*b*) That as the area of breaking-surface increases towards the discharge end, the material as it is broken smaller is spread over a larger space, which results in finer crushing than is practicable with a reciprocating jaw-machine.

(*c*) That as the head is loose on the eccentric spindle, there is no grinding action, and there is a line of crushing-surface from top to bottom of the head and liners in constant action.

(*d*) The area of breaking-surface is about three times as great as in reciprocating machines of similar power. This seems to admit of the machine doing nearly treble the work in any given time, driven at the same speed, and of tipping into it large quantities of stone (using a hopper); dispensing with much of the attention necessary with machines belonging to class A.

(*e*) That, having a comparatively small crushing-surface acting continuously, dispensing with a fly-wheel to store up power (part of which is wasted), and the rock which is to be broken being supported at its ends only, in the circular crushing-chamber, less power is required, as compared with a jaw machine, to do a given amount of work.

Quoting the report of a committee, on a competitive trial between a 9 inches by 15 inches Blake-Marsden, and a 9 inches by 14 inches Gates, at Meriden, Connecticut, on May 30th, 1883, the makers of the Gates assert that, crushing the same kind of stone to an equal size, the latter machine crushed over three times as much in a given time, and showed a saving of about 33 per cent. of the indicated power consumed in doing the work.

On the other side, it must be said, that a Gates of an equal capacity will not take in a lump of rock of as large sectional dimensions as a Blake or Marsden; and the latter machines have the advantage in respect of weight and prime cost; the price of the large sizes is in fact extremely heavy, and the height of the Gates is also a disadvantage, where fall has to be economized. The injury done to the machine, which might result from a hammer head, or drill falling into it (a not unfrequent occurrence through carelessness, or occasionally other reasons) has also to be considered, and there is no doubt that the Gates requires particular attention, in keeping it properly lubricated with suitable lard, or heavy lubricating oil. The head and liners are made of specially chilled, cast white-iron.

There is one particular case, however, which should be pointed out, in which the Gates seems to possess very special advantages over other crushers, and that is, where there is a wide lode yielding large quantitie

of milling ore, in which the stone breaks "large" in stoping it down, and requires in consequence to be spalled, to get it into the jaws of an ordinary breaker. Under such circumstances, a Gates may obviously serve a very useful purpose indeed, as the larger sizes will take in whole lumps, as big as can be handled, and by this means the tedious and expensive process of breaking in the mine, or "breaker-floor," is avoided.

The measure of economy in using a large sized breaker of the kind will necessarily depend, however, on being able to keep it fully employed, as well as possessing ample storage-bin room and driving power. As an instance of the work it can accomplish under such circumstances, the Caledonia mill in Dakota may be quoted,* where one No. 6 Gates, with receiving-openings 12 inches by 18 inches, attended by one man, crushed 200 tons in 10 hours, and did the work formerly set for three No. 5 Blakes (the largest ordinary pattern in the market), with a receiving-opening 9 inches by 15 inches. These latter took five men 20 hours, it is said, to produce the same result, requiring the same amount of power. It is asserted by the makers of the Gates, that its action entails less stress on the structure it is mounted upon, than is the case with a reciprocating breaker, but one is somewhat inclined to doubt whether this holds good in starting it when empty, whatever may be the case whilst it is running full.

Its size, and weight, must evidently be special objections, when it has to be mounted on a lofty structure (like the back floor of a stamp mill); and where the stone is divided, along a long line of bins, the advantage of using one large machine, is more or less offset by the difficulty of distributing the ore between them, unless there is an exceptionally large fall to spare. Under such circumstances the only practical escape from the difficulty, is to set the breaker apart from the mill (if possible close to the shaft); and it is not improbable that this is, as a rule, (where the mine and mill are owned by the same company, and the former keeps the latter entirely occupied), the best disposition to adopt. This does away with much of the dust, which plays havoc with the bearings of the other machinery of the mill, permits of lighter framing for foundations and superstructure, less grading (in some cases),† and a smaller annexe to the main building.

2.—*The Lowry Ore-breaker.*—A second form of gyratory crusher has quite recently been introduced by the National Machinery Co., of Tiffin, Ohio.

* "Gold-milling in the Black Hills," by H. O. Hofman. *Trans. Am. Inst. Min. E.*, vol. xvii., page 498.

† When rock-breakers and ore-bins are used with automatic-feeders at the back of a stamp-battery, which has tables and concentrators below it, the least practicable fall that ensures the automatic movement of the ore through the various stages of the process is about 33 feet from the rock-breaker to the concentrator floor.

A description of this crusher appeared in the *New York Mining Journal* of December 5th, 1891.

Its construction is illustrated by a diagram, which shows it in section, with a removable revolving top for fine-crushing, containing two or more holes, to regulate and spread the stone round the breaker-head. The ore is thrown into a hopper, and is discharged as fast as it is broken, avoiding any danger of clogging. The Lowry ore-breaker appears to occupy the same position with regard to the Gates, that the Dodge does to the Blake, as the vertical shaft is fulcrumed below the crusher-head instead of above it; consequently there is less motion, so it is asserted, at the bottom of the jaw, and the product is said to be more even than that of any of the jaw-crushers, in which the jaw is pivoted at the top, a statement which is no doubt correct.

The chilled-iron head or cone appears smaller in the Lowry breaker than in the Gates. The chilled-iron ring or liner, which forms the outer crushing-face is cast in one solid piece, having a solid turned bearing and supporting-backing behind it. The machine is adjusted for crushing to various sizes by screws at the side of the crushing mortar, by which the liner can be raised or lowered. This is done to obviate the necessity of lifting the vertical shaft, carrying the crusher-head, the bottom of which always remains in the oil chamber of the eccentric, at the base of the machine. The vertical shaft is driven by ropes with sufficient laps round the pulleys, and with provision for adjustment in the tension-pulley.

As one of the chief advantages of this machine, the manufacturers claim that the crusher-head strikes downward, and has a tendency to draw in the material to be crushed, which would not be the case if the vertical shaft were fulcrumed at the top.

These breakers are made in nine sizes, having a capacity varying between 5 and 250 tons per hour.

Without having seen both machines at work and knowing more about the Lowry, it is impossible to institute a fair comparison between it and the Gates. But it is probable that, comparing the former with reciprocating-jaw machines, most of the arguments for and against the Gates seem to apply equally to its new competitor.

The purpose for which an ore-breaker is intended, its capacity, saving in space occupied, weight, quickness in setting to work, ease of repair, simplicity of construction, wear per ton of ore, liability to serious or minor accidents, first cost, and consumption of labour in attendance, lubricants, and power required per ton of ore crushed, will all influence the choice of a machine, and no hard-and-fast line can be drawn, between the

various coarse-crushers which have been mentioned, each possessing a sphere of usefulness of its own. In fact, as some of the above conditions are to a certain extent conflicting, a careful consideration of the particular circumstances of each case is absolutely necessary. In judging of the relative economy of different crushers in regard to consumption of power, it is perfectly correct that the only true test of their efficiency is the actual, indicated, useful effect shown per lb. of fuel burnt in cubing a certain class of stone to a given size. This method of comparison is, not however, so entirely reliable as would at first sight appear, unless the machines under trial be driven with the same engines and boilers, by the same enginemen.

The consumption of fuel will, in fact, differ in different works, according to its quality or description, the size and type of engine and boiler employed, and the care bestowed in looking after them.

Further than this, it is absolutely necessary to compare the exact time the crushing lasted, deducting all stoppages, since an engine which is idle, during the time the boiler fire is alight, may be consuming—in proportion to the useful work done—much more fuel than either the indicated or real horse-power (measured while the engine was working) would lead one to suppose. The different sizes, weights, capacities, size to which the stone is cubed, size and speed of driving-pulleys, nominal horse-power expended, and cost of the Blake, Blake-Marsden, Dodge, Gates, and Lowry ore-crushers, are given in the subjoined tables, as far as it is possible to ascertain them.

DIMENSIONS, PRICES, AND PRODUCT OF THE BLAKE STONE- AND ORE-BREAKER.*

No.†	Size of Receiving Opening.	Approximate Product per Hour in Tons crushed to 2 inch size	Weight of Heaviest Piece.	Total Weight.	Extreme Dimensions.						Driving Pulley.				Proper Speed. Revs. per Min.	H.P. Required.	Price F.O.B. Chicago.		
					Length.		Breadth.		Height.		Diam.		Face.						
	Inches.		Lbs.	Lbs.	Ft.	In.	Ft.	In.	Ft.	In.	Ft.	In.	Ft.	In.			£	s.	d.
...	3 × 1½	Laboratory	40	100	1	1	0	6	0	10	0	5	0	1	300	...	10	8	4
A	10 × 4	4	1,993	4,719	4	6	4	0	3	11	1	8	0	6	...	4	62	10	0
2	10 × 7	6½	3,950	7,896	5	4	4	6	4	5	2	0	0	7½	275	7	104	3	4
5	15 × 9	10½	7,179	15,749	6	8	5	0	5	3	2	6	0	9	...	10	156	5	0
8	20 × 10	13½	7,700	17,000	6	10	5	9	5	11	3	0	1	0	250	14	218	15	0

* Messrs. Fraser and Chalmers' *Catalogue, No. 4* (Gold and Silver Mills), and Egleston's *Silver*, page 358.

† There are various intermediate sizes not mentioned in the table; the 20 inches by 15 inches, with a product of 20 tons per hour, being the largest.

Size A—Takes pieces 4 inches thick, and crushes to ½ inch or less. 2—Takes pieces 7 inches thick, and crushes to ¾ inch or less. 3 and 4 are used for road-ballast, furnaces, and breaking smaller for other crushers. Mr. H. S. Munroe* places the cost of repairs for eight 7 inches by 15 inches lever-pattern crushers (Blakes) with corrugated jaws, estimated to have crushed 224,203 tons of limestone, at Bonne Terre, Missouri, in 12 months, at:—

	£	s.	d.
12 levers, at $25·00	62	10	0
9 jaw-plates, at $15·00	29	1	3
12 jaw-plates, at $12·00	30	0	0
Toggles, check-plates, and sundries ...	51	12	6
Total cost	£173	3	9

This represents an average of about £20 16s. 8d. for each crusher, breaking 80 tons per diem, to 1½ inches, but does not include the cost of babbitting bearings, or labour in making repairs.

From these data, the average life of the wearing parts of a jaw-crusher, appears (similarly circumstanced) to be about 8 months.

The lower box of the pitman-shaft must be packed with thin wood packing, to prevent the key tightening the box to the shaft. Iron plugs should be kept in the oil-holes, and if the fixed jaw requires to be cast-up, it should be backed with zinc plate about ¼ inch thick.

The amount of product depends on the distance the jaws are set apart, the speed, and also on the nature of the ore.

The product given in the table assumes the jaws set 1½ inches open at the bottom, the machine run at proper speed, and properly fed, but it will vary somewhat with the character of the stone, brittle stone for instance going through faster than sandstone.

The No. 2 Blake is made in sections for mountain transport. Total weight, 6,881 lbs.; cost, £145 16s. 8d. No piece of this machine weighs over 332 lbs.

DIMENSIONS, PRICES, AND PRODUCT OF THE DODGE CRUSHER.†

No.	Size of Jaw Opening.	Approximate Product per Hour to Nut Size.	Total Weight.	Driving Pulley. Diameter.	Width of Belt used.	Proper Speed. Revolutions per Minute.	H.P. Required.	Price F.O.B. Chicago, Complete.
	Inches.	Tons.	Lbs.	Inches.	Inches.			£ s. d.
1	4 × 6	½—1	1,200	20	4	275	2—4	52 1 8
2	7 × 9	1—3	4,300	24	5	235	4—8	93 15 0
3	8 × 12	2—5	5,600	30	6	220	8—12	114 11 8
4	10 × 16	5—8	12,000	36	8	200	12—18	187 10 0

* "The New Dressing Works of the St. Joseph Lead Company." *Trans. Am. Inst. Min. E.*, vol. xvii., page 659.

† Messrs. Fraser and Chalmers' *Catalogue, No. 4.*

DIMENSIONS, PRICES, AND PRODUCT OF THE BLAKE-MARSDEN (WITH LEVER AND ECCENTRIC MOTION) STONE-BREAKERS.*

Size of Receiving Opening.	Approximate Product per Hour to Road Metal Size, 2½ Inches.	Total Weight of Machine.	Extreme Dimensions.			Driving Pulley.		Proper Speed. Revolutions per Minute.	H.P. Required.	Price.
			Length.	Breadth.	Height.	Diam.	Face.			
Inches.	Tons. Cwts.	Lbs.	Ft. In.	Ft. In.	Ft. In.	Ft. In.	Ft. In.			£
12 × 8	5 8	10,080	8 1	3 7	4 9½	2 0	0 5	250 to 300 revolutions.	5	113
15 × 8	6 5	12,880	8 9½	4 10	5 3	2 6	0 6		7	135
15 × 10	7 10	15,680	8 9½	4 10	5 3	2 6	0 7		8	150
18 × 9	8 15	17,584	8 11	4 11	5 0½	3 0	0 7		9	165
20 × 10	10 0	22,400	9 11	5 5	5 4½	3 0	0 7		10	180
24 × 13	15 0	30,240	10 6½	5 8½	6 7½	3 0	0 9		12	263
24 × 16	16 0	31,360	10 6½	5 8½	6 7½	3 0	0 9		14	275
24 × 17	16 5	31,920	10 6½	5 8½	6 7½	3 0	0 9		14	282
30 × 18	20 6	33,600	...	...	...	...	...		18	375
34 × 18	23 2	42,560	...	...	...	...	...		20	413

The makers state that it is advisable to use an engine of 2 horse-power more than stated in the table, as it is more economical to run a larger engine, than to run one up to its full power (this remark also applies to the Blake and the Gates crushers). The principal working parts are all crucible steel.

DIMENSIONS, PRICES, AND PRODUCT OF THE GATES OR COMET ORE-CRUSHER.

Size.	Dimensions of the Three Receiving Openings combined.	Dimensions of each Receiving Opening, about.	Weight of Breaker.	Capacity per Hour in Tons of 2,000 Lbs. passing 2½ Inches Ring according to Character of Ore.	Dimensions of Driving Pulley.		Space occupied by Breaker set up.			Diameter of Hopper.	Revolutions of Pulley.	H.P. of Engine recommended to drive Breaker, Elevator, and Screen.		Net Price F.O.B. London.
					Diam.	Face.	Height from Bottom Frame to Top of Hopper.	Width of Frame.	Length of Frame.			Limestone.	Granite.	
	Inches.	Inches.	Lbs.		In.	In.	In.	In.	In.	In.				£
00	2 × 12	2 × 4	500	Laboratory and sampling	8	2⅜	24	17	26	13	700	½	½	25
0	4 × 30	4 × 10	3,100	2—4	16	6	48	30	73	28	500	4	4	85
1	5 × 36	5 × 12	5,500	4—8	20	7	54	31	76	37½	475	8	8	125
2	6 × 42	6 × 14	7,800	6—12	24	8	60	39	90	39½	450	12	15	165
3	7 × 45	7 × 15	13,500	10—20	28	10	73	48	103	44½	425	20	30	250
4	8 × 54	8 × 18	20,000	15—30	32	12	85	54	114	51	400	30	40	400
5	10 × 60	10 × 20	27,000	25—40	36	14	96	63	123	59	375	40	50	500
6	11 × 72	11 × 24	36,000	30—60	40	16	109	73	139	66	350	50	60	725
7½	13 × 90	13 × 30	60,000	40—75	48	18	116	73	144	120	350	60	75	1,040
8	18 × 135	18 × 45	89,000	100—150	48	20	156	90	164	132	350	125	150	1,450

* Makers' catalogue and correspondence.

The power required per ton will, of course, vary with the size to which the stone is broken, requiring additional power, to break it smaller than ordinary macadam.

An elevator and screen are used when it is desired to break everything down to $\frac{3}{8}$ inch.

If the product is not required smaller than $\frac{3}{4}$ inch, and some "spalls" are not objectionable, the screen and elevator are not needed.

Prices of Gates' breakers with revolving screen and return-elevator fitted complete, on frame; size 0, with 24 inches by 36 inches revolving screen, £156; size 1, with 24 inches by 36 inches revolving screen, £208; size 2, with 24 inches by 36 inches revolving screen, £250; size 3, with 24 inches by 36 inches revolving screen, £354.

Elevators 24 feet between centres, including connexions, range in price for the various size breakers (from 1 to 7) from £56 to £127, and weigh from 2,000 to 5,000 lbs. Revolving screens, with dust jackets, cost from £42 to £146 (exclusive of connexions) and weigh 2,000 to 8,000 lbs.

Medium-coarse and Fine-crushing.

From the point of view of the mechanical engineer the superiority of one machine of class II. over another, for the reduction of a certain kind of stone (say granite), can in a measure, be pretty easily demonstrated, but the question as a miner requires to look at it, becomes one of extreme complexity, as the factors to be dealt with are variable and conflicting, in more ways than one.

Viewed in this light, if the field of selection be limited, merely to the three classes of crushers particularized on page 1 the choice of the best for any special case, appears to depend on three equally important considerations, which are altogether outside the primary question of the mechanical merits or demerits of a particular machine, and it is here that the experience of the mining engineer is called into play.

To arrive at a trustworthy solution of the problem, he must take into consideration:—1st, the character of the ore itself and its associated gangue; 2nd, the degree of comminution to which it is desirable to carry reduction in order to get the best results out of any particular system of treatment that may be adopted; whilst in some cases, the location of the mine and the nature and position of the mill site, also form a 3rd factor of the question.

To put the matter in a nut shell and go a step farther, it may be laid down as a rule, that setting aside any other machines which have claims to attention, where one of the three classes of crushers (stamps, rolls, and roller-mills) now under consideration is found, commercially speaking, to best meet the needs of the miner in any one instance, it is most unlikely that either of the other two types, will be found suited to the requirements of the same case; if indeed, they did not turn out on trial absolute failures, supposing one kind of machine to be substituted for another.

An ideal "reduction plant," would have to establish a claim to extract the largest percentage possible of gold or other metal automatically in the shortest possible time, and in the simplest possible manner, combining capacity for expeditiously reducing the greatest quantity of ore in a given time to any desired point of fineness, with a minimum production of fines or slimes; it should further possess strength and durability without being cumbersome, and have as few working-parts and auxiliary appliances connected with it as possible. Now, while this acme of mechanical and metallurgical perfection combined has yet to be invented, the success of one type of crusher as against another in various instances, may be directly traced to the fact that some of those features which are wanting in one class of machine, are found more or less developed in one or other of its competitors. In selecting the most suitable machinery for his own particular object it is therefore incumbent on the miner not to lose sight of the points that have been indicated, as they may relatively affect in a different degree the circumstances he happens to have to deal with.

The adaptability of the plant in the foregoing respects, however, is not everything, as the outlay it may involve must after all be justified by the returns it will yield. He has therefore to compare:—

1. The prime cost and weight of each class of machinery, with its indispensable adjuncts for extracting the commercially valuable metals from any particular ore, representing a very variable capital outlay to produce a given result.
2. The simplicity or complexity of the entire plant in each case. not only with regard to cost of repairs, but also the time lost in maintaining it in running order in out-of-the-way places. difficult of communication, and where skilled labour is perhaps not available.
3. The relative time required to erect each different class of machinery; and lastly,

4. The comparative costs (both in regard to quantity and price) of labour, fuel, oil, etc., required to run a given class of plant in a certain locality.

Under these circumstances, with the limited data possessed at present in the shape of independent figures throwing light on many of these questions of relative efficiency and cost, at most only general statements can be made; and any attempt to assert the adaptability or superiority of one machine or class of machines for all cases, must infallibly break down.

Each form of crushing machinery has its champions and detractors who freely express themselves for or against it, according to the measure of success it happens to have achieved merely in their own limited personal experience; and where failure occurs the machine not unfrequently is blamed, when in some instances it is owing to want of forethought on the part of the purchaser in overlooking natural contingent results, which might have been reasonably foreseen and easily escaped.

The degree of comminution to which it is desirable to carry reduction to get the best results out of any particular system of treatment rests on the general nature of the process employed to extract whatever of value the ore contains; this is dependent to a very large extent on the ore itself and its associated gangue.

The choice of medium and fine-crushing machinery must then be regulated, from a mechanical standpoint, as well as by the class of ore it is required to crush, bearing in mind that ruinous losses may ensue by going to either extreme. That is to say, in selecting a machine, if crushing is carried to an unnecessary degree of fineness, money is wasted in doing unnecessary work, which shows itself in the cost sheet at the end of the year, besides risking various mechanical and metallurgical losses.

Amongst these latter, may be enumerated, slimes and fine mineral in suspension, in concentration, and wet amalgamation; dust in furnace and other dry processes; and various inconveniences in filtering and extraction, in those cases where ore has to be lixiviated; all entailing extra expense, to mitigate a difficulty which has been artificially created.

If, on the other hand, the crushing is not fine enough, a proper percentage of the metals (remaining as they do locked up in their stony envelopes) is not secured. So that whether water be employed to concentrate them, mercury to entrap them, chlorine to attack them, or some chemical solution to extract them, there is a more or less considerable loss, which ought to be carefully studied, for the purpose of finding the

vanishing point (so far as it can be learnt) common to all these combined sources of trouble.

One may say then, that there is a certain average degree of fineness, to which it pays best to carry reduction in every process (varying with the ore, and the method of treatment). But before this point can be settled it is necessary to decide what is the best process for the purpose in view, *i.e.*, in getting the largest commercial return on a given capital outlay.

On these grounds, it is of paramount importance that the chemical and physical peculiarities of the ore and its associated gangue should be first studied, and carefully noted in connexion with the local surroundings of the works, in regard to the character and cost of labour, fuel, supplies, and abundance or absence of water-supply, as well as climate and other considerations, which must influence the choice.

To enter into minute details concerning the peculiarities of various ores, and the processes of treatment they might be subjected to, is far beyond the scope of this paper, but from a general standpoint, as affecting the present question, the materials (ore and country-rock and gangue) which the miner, the metallurgist, and the mechanic, have alike to deal with, may be divided according to their physical characteristics, in three chief classes :—

(*a*) Brittle and hard; such as quartz, most pyritic ores, syenite, and coarsely crystalline metamorphic rocks.

(*b*) Tough ; such as native copper, gneiss, and most of the micro and crypto-crystalline metamorphic and intrusive volcanic rocks, which accompany ore-deposits.

(*c*) Soft or clayey ; these latter comprising earthy carbonates, surface ores, or clayey limestones, as well as some classes of stone, comparatively hard and brittle like schist but carrying clay-partings, and sandstone.

When the different physical peculiarities which attach to the first elementary mineral substance in the above list are considered, it is seen what wide room there may be for enquiry in this direction alone.

Silica which possesses no doubt a special interest to many engineers when it takes the form of gold-quartz, may possess under certain circumstances, what miners term a kindly appearance, tinged sometimes a light-grey, blue, or brownish-red colour, combined with a dark resinous look; whilst elsewhere it may have a vitreous, wet, or opaque-white, hungry appearance, probably so-called by "cousin Jack," because it generally leaves him (if he attempts to work it) precisely in that very condition.

Still some of the best specimen-quartz is frequently milky, and white as a hound's tooth, whilst what is known as ribbon-quartz, a laminated variety of stone (separated by thin talc or chlorite-partings, or laminæ of graphitic shale or slate) is not unfrequently when met with, a good weight-carrier of metal. All these physical differences, due to peculiarities of hydration, association, and structure, will influence the way in which the stone breaks, and therefore affect its after treatment.

The materials dealt with by the mining engineer may, therefore, be divided according to their mineralogical characteristics and metallurgical behaviour, into a further number of classes, with different subdivisions:—

(*a*) Ores of the base metals, whether containing gold or silver or not, which have to be smelted or treated by a wet process, either with or without a previous (hand or mechanical) coarse or fine concentration, matting, or roasting, depending on circumstances which need not be enumerated.

(*b*) Ores of one or both of the precious metals, properly so-called, comprising:

1. Free-milling gold ores.
2. Free-milling silver ores.
3. Pyritic ores of gold and silver.
4. Combinations of 1, 2, and 3.
5. Exceptional ores.

These different kinds of ore may be subjected to various processes, amongst the most general of which, for dealing with free-milling* low-grade gold ores (belonging to class 1) are:—

1. Stamping the ore wet to fine size, and catching the free gold on copper plates (inside and outside the battery), with wells and mercury traps, used as accessories.
2. Wet stamping the ore coarse, and amalgamating on copper plates, separating the coarse-sands from the finer slimes, by screens or sizing-boxes (in some cases, preceding or following this separation by fine concentration); and regrinding the coarser sands thus saved, in Huntington or Chilian mills, or pans;† followed by more copper plates.

For dealing with free-milling comparatively high-grade ores of gold, belonging to class 1, in which there is an excessive loss, if the previously described systems of treatment are followed (owing to the fine

* A term implying that the greater part of the gold is free.

† Without mercury preparatory to re-amalgamation.

state of division of the metal, its alloyage with silver, or some other reason) there may be employed :—

1. Wet stamping, succeeded by direct pan-amalgamation on the ordinary plan (or else what is known as the Ross continuous system); or crushing dry (followed in exceptional cases by roasting) and then amalgamating in Chilian mills or pans. When there is a small percentage of rich sulphides present (especially if dealing with a mixed gold and silver ore) fine, *i.e.*, table-concentration, may precede or follow the pan-treatment with an appreciable saving.
2. In a few special cases, like the Mount Morgan mine, Queensland, ores of this class may no doubt be treated in bulk with advantage by drying, crushing with rolls, partially roasting, and working by chlorination in barrels. That the Mount Morgan ore may equally well be classed amongst the exceptional ores (belonging to class 5), becomes evident however, when the facts of the case are closely studied.

Mr. T. A. Rickard in a paper read before the American Institute of Mining Engineers, June, 1891, says :*—"The complete success of the treatment is largely due to the extreme friability of the ore, which renders its pulverization easy, whilst its porosity assists materially in the thorough chlorination of the gold." An enthusiastic writer has spoken of the ore as a sort of snow-drift, which melts in the chlorination vats of the company, into a golden sand, such as might be supposed to have been brought from the bed of the river Pactolus, instead of from the top of an Australian mountain. The extreme richness of the stone hitherto available for treatment in large quantities (the capacity of the works being 1,800 tons per week), the extremely minute state of subdivision of the gold, and the physical peculiarities of the ore above referred to, have all tended to make barrel-chlorination applied in this peculiar way a success, where the stamp-mill has been unsuccessful, and this also accounts in no small degree for successful crushing of the ore with rolls.

Mr. Rickard remarking on the cause of failure when the ore was stamped, says :—"The rock which the battery (25 stamps at Dee Creek) was called upon to crush, averaged 10 ounces per ton, but the contents of the tailings proved of much greater value than the amount of amalgam obtained. This led to a critical examination of the ore at the Sydney Mint." It was found that the bullion was of a fineness hitherto unknown in nature, assaying 99·7 per cent., occasionally 99·8 per cent., of pure

* *Trans. Am. Inst. Min. E.*, vol. xx., page 150.

gold, the rest being copper with a trace of iron. It is remarkable as being almost entirely free from silver. Dr. Lebius, of the Sydney Mint, considered as a result of numerous experiments, that the iron present was in the form of an oxide, which coated the gold, and so prevented its contact with mercury. In a footnote Mr. Rickard adds: "My own experience with the ores of Gilpin County, Colorado, leads me to believe that this is more frequently the obstacle to successful amalgamation than is usually supposed.

For dealing with free-milling silver ores (carrying less than 6 ounces per ton of white metal) belonging to class 2 as a rule one may employ :—

1. Wet stamping, followed by direct amalgamation in pans or barrels. The ores of this class are limited, only embracing decomposed surface-ores, carbonates, and occasional deposits of argentite, chlorides, chloro-bromides, and native silver.

For dealing with pyritic ores of gold and silver, belonging to class 3 which contain the larger proportion of their metallic contents in combination with sulphides. The following are the usual methods :—

(*a*) If the sulphides are massive.

1. Hand-picking and after treatment, as described below.

(*b*) If the sulphides (pyrites) are disseminated.

2. Wet stamping, combined with battery and plate-amalgamation, to extract the free gold followed by fine concentration of the pyrites, and treatment of the hand-sorted ore and concentrates by smelting for lead, which yields bullion that has to be refined by different processes; or by smelting for copper in reverberatory furnaces, to produce an enriched copper matte.* The choice of these methods depends on whether lead or copper is the predominating metal. More frequently, however, the gold is extracted from the concentrates by the ordinary Plattner vat-chlorination treatment, whilst in a few special cases an iron matte is produced, and sold to smelters, or crushed and chlorinated.
3. Wet stamping, followed by amalgamation and fine concentration, and grinding the concentrates raw, in pans or Chilian mills.
4. Dry stamping, followed by roasting, with or without salt, and amalgamation in pans or barrels, a process which is only adapted, however, as a rule, to rich silver ores, carrying a certain proportion of sulphides, but not enough to pay for concentration.

* The gold and silver in this matte, are subsequently extracted by a wet process from the copper-bottoms which contain the gold, after the silver has been extracted by the Ziervogel or Augustine processes.

5. The new cyanide-lixiviation-process, which is claimed to be applicable to the treatment of certain gold ores, much in the same way as the Russel process is to silver.
6. Ordinary lixiviation.

Dealing with ores belonging to class 4, which are amongst the most difficult to treat, because a process adapted to the extraction of gold may be very inefficient for the recovery of silver, and *vice versa:* when one metal is in small proportion as regards value, it is frequently sacrificed to the most profitable process commercially for the other. Many of the base silver ores of the United States contain 5 to 15 dwts. of gold, and from 20 to 50 ounces of silver per ton. For such ores dry stamping, roasting, and pan-amalgamation is mostly used, and under proper conditions will extract 90 per cent. or more of the silver, but only 40 to 60 per cent. of the gold. Lately, however, coarse-crushing, instantaneous roasting in Stetefeldt furnaces, and double-leaching by the new Russel process has come into prominence, and is said to be giving excellent results in certain instances, as regards economy and extraction of both metals.

The Russel process seems to be more particularly applicable to those ores which do not contain enough lead or copper for smelting, are poor both in gold and silver, and which contain large amounts of sulphur, arsenic, and antimony, since roasting with salt would convert the base metals as well as the silver into chlorides, and would give a very base bullion if it were amalgamated. There is, however, a remedy for this, which Mr. McDermott points out*: "When the silver is in combination with sulphides, antimonides, arsenides, and tellurides of the baser metals, the pan process becomes inefficient and expensive. By contact with iron surfaces, heat, and the addition of some chemicals—chiefly salt and sulphate of copper—a partial decomposition of the complex minerals is effected, and some of the silver amalgamated, but the wear of iron, loss of mercury, cost of power and chemicals, and the production of base bullion together, go far to neutralize the gain made in the recovery of the silver. In such a case a great benefit is derived by combining concentration with amalgamation. The light, flocculent chlorides and sulphides of silver can be amalgamated to a high percentage, while concentration is almost useless to deal with them in the form in which they exist in a free or decomposed ore.

On the other hand, concentration can be made very effective on the undecomposed complex minerals, for which amalgamation is ill-adapted.

* *Gold Amalgamation*, by Walter McDermott and P. W. Duffield, page 88.

There are two methods of combining the processes according to their order, concentrating either before or after the pan-treatment. In regard to the relative advantages of the two, concentration before amalgamation is the natural method, because it relieves the pans of the baser minerals, which are a disadvantage in the amalgamation, and the subsequent concentration of which is made more difficult by the grinding or attrition of the minerals in the pans.

The only argument against the universal adoption of this order, rests on the disadvantage of sometimes having native metals, and some chlorides and sulphides entering the concentrations, instead of appearing at once as bullion, which they otherwise would do, and also, that very perfect settling of the slimes from the concentration tails is necessary, to prevent loss of the flaky silver chlorides and sulphides. This process is in use at the Standard mill, California.

When the free metal is gold, the first-named disadvantage can be overcome by using copper plates before the concentrators, and this process was adopted by the Montana Company with great success, after first trying pans before the concentrators.

Comparing the combined process with dry crushing and roasting, the advantages of the former consist in a reduction of the cost of working fully one-half of that of dry crushing, a crushing capacity of double for the same number of stamps, a decreased cost of erection, and a higher saving of gold present. Against these advantages may be placed simply the increased percentage of silver saved by the dry process under certain circumstances.

When combined gold-and-silver ores carry over 10 per cent. of base metals it usually happens that the silver does not exist equally in the minerals present, but is concentrated in one of them as a rich, brittle ore. It follows from this, that concentration in some instances is very ineffective, because while 90 per cent. of the heavy base metals may be saved in the concentrates, the loss of the fine silver-bearing mineral in the slimes may be fully 50 per cent. of the assay-value of the crude ore. In some cases the clean concentrated, heavy minerals do not assay more than the original ore from which they were separated.

This is a very strong argument against the injudicious application of fine concentration to all cases, for, as Mr. McDermott says, there are ores of both gold and silver in which the sulphides are so rich, that even a very small loss by weight involves a very large loss by assay of the precious metals.

Such minerals as tellurides of gold and silver, ruby and brittle silver

ore, come under this head. In concentrating some of the tellurium ores of Colorado, some of the very finest slime-material, overflowing from the concentration-tanks of the Frue vanners, assayed as high as £5,000 per ton in gold and silver.

Where there is danger of loss, as described, it is advisable to introduce a double concentration, re-treating the tails on vanners.

For dealing with exceptional ores belonging to class 5: if their value lies mostly in tellurides of gold and silver, with little or no sulphides present, the most successful method yet found is by selection of the rich ore; and concentration (as above described), by double treatment of the low-grade ores. Both rich ore, and concentrates, are then so valuable, that smelting on the spot or shipment to smelters is advisable.

Another class of exceptional ores are those, in which the gold is not free, so far as tests indicate, and yet there are no sulphides present.

These ores are generally oxidized in character, changed from their original form, and contain chloride of silver in addition to the gold. As some of these ores, when leached with hyposulphite of soda, yield a large portion of the gold as well as the silver, it would seem to indicate that the gold exists as a chloride, combined with the silver, either as a double insoluble chloride, or mechanically protected by the latter from being leached out by the natural drainage-water of the mine. Such ores are uncommon, but have been worked to some extent by a raw-leaching and concentration for the fine carbonate of lead, and its combined silver which is also present.

There may be, other explanations, however, to account for the peculiar character of the ores just alluded to. On page 6 of *Gold Amalgamation and Treatment*, the losses of gold in milling are referred to five principal causes:—

1. Loss of free gold, quicksilver, or amalgam, due to carelessness or inexperienced amalgamation.
2. Free gold and gold-bearing sulphides, attached to or embedded in particles of rock.
3. Gold contained in base-metal sulphides, broadly termed sulphides.
4. Gold lost in fine slimes, and sometimes in solution.
5. A condition of gold in which it is not susceptible of copper-plate amalgamation.

It is with the last of these that we now have to deal.

Quite recently * Mr. Richard Pearce, of Argo, Colorado, has given

* "The Association of Gold Ores with other Metals in the West." *Trans. Am. Inst. Min. E.*, vol. xviii., page 447.

some most interesting facts with regard to the association of gold with other metals in Colorado, which bear on amalgamation and concentration. His investigations go to prove that gold and silver exist in some of these ores found below the water-line, alloyed with bismuth, tellurium, and copper, and perhaps arsenic, where the presence of such combinations has hitherto been unsuspected, and a coarse-grained almost pure pyrites containing silver and a small value of gold, from the Louisville mine of Leadville, has been proved to contain metallic tellurium. When thus combined the gold would appear to be more or less refractory to the ordinary milling treatment, and this may account for some of the troubles set down to so-called rustiness.

A natural crystalline compound of gold and bismuth, has lately been discovered in Australia, it is said, and native alloys of these metals are known under the names of maldonite and bismuthaurite. Hessite, a natural combination of tellurium and silver (Ag_2Te) has been found in some of the Red Cliff veins, and a telluride of bismuth, probably tetradymite was found by Mr. Pearce in ore discovered at Ouray. He contends that, in certain cases, bismuth performs, a by-no-means small part, in influencing the general deposition of gold, and also in causing the refractory behaviour of this metal under metallurgical treatment.

Mr. Pearce adds: "The occurrence of gold in this district seems to be intimately associated with that of eruptive porphyries, and the impregnation of veins with gold-bearing minerals is apparently always accompanied with silica. Evidence of intense thermal action* is one of the chief characteristics of gold deposits. Examples of these processes may be found in Gilpin, and also in Boulder County. In the latter district known as the 'telluride Belt,' there are indisputable evidence of vein-impregnation by the circulation of siliceous waters through the joints of the porphyry; in this case tellurium is the chief mineralizing agent of the gold. Perhaps the most striking example of the deposition of gold, through the agency of thermal waters, may be seen in the celebrated Bassick mine, which has so often been described. This deposit exhibits all the characteristics usually accompanying geyser action. The disintegration of the porphyry, with a partial replacement of the felspar by silica, is here clearly shown. The gold was combined almost in its entirety

* The idea Mr. Pearce intends to convey (which is the writer's view of the matter) is that the filling of veins with mineral, is rather due in most cases to the leaching of the adjacent rock-masses, followed by a gradual replacement of the original constituents of the lode with ore, along certain horizons or belts, than to the action of thermal waters, building up masses of mineral in an open fissure, though both solfatara action, as Mr. Pearce remarks, at the Bassick mine, and sublimation in exceptional instances, may have tended to such a result.

with tellurium, and rich tellurides of gold and silver were found sprinkled through the mass of material. In some remarkable specimens these tellurides formed a distinct coating on the surface of smooth boulders which had become rounded by attrition from the action of steam."

Dr. T. Egleston, in an elaborate paper on the causes of rustiness and some of the losses in working gold ores,* points out that the term rusty is used by miners to indicate a condition in which the metal is supposed to be coated superficially or alloyed with some substance, which prevents contact with mercury, and consequently precludes the possibility of amalgamation.

Gold in this state, it appears, is frequently covered with a brownish coating, which has a much redder colour than ordinary gold, and is irregularly distributed over its surface: where the least abrasion has occurred the metal underneath show the true gold colour. Fine particles of gold are sometimes visible with the microscope in the detached coating, and it often cracks off from pieces of gold, leaving them bright. Very often this film is composed entirely of silica, deposited on and beside the gold. It is sometimes opaque, and again quite transparent, so that the gold can be seen with the microscope disseminated through it, just as cinnabar-crystals are seen in the red chalcedony of the district around Knoxville, California. There may be, for example, many artificial causes which produce this rustiness of gold, the covering of the surface with particles of some foreign substance, or its alloyage with other metals, as already explained.

As Mr. McDermott remarks, concentration and pan-amalgamation would probably be effective on the former class of ores, though for reasons that will presently appear the writer would like to add, or concentration preceded by crushing in centrifugal roller-mills, if other circumstances admit of it. Dr. T. Egleston's view is, that the action of rubbing, which occurs in any machine like the arrastra, is much more likely than the stamp-mill to pulverize the fine pyrites, break up any coating that may be around the particles of gold, rub off the superficial deposit, and thus bring the gold into contact with the mercury and make it amalgamate. He states, as a remarkable fact, that in the early days, Mexicans with the arrastra, got 50 to 60 dollars a day, where stamp-mills, working the same rock, only obtained 15 to 20 dollars, and instances can be cited where, with the best modern machinery, only 20 to 30 dollars can be got out of rock which yields 700 to 800 dollars by fire assay.

* *Trans. Am. Inst. Min. E.*, vol. ix., page 646.

Of course this might be due partly to the first cause of loss cited, viz., carelessness or inexperience on the part of the workmen running the mills, but this is on the whole unlikely. Amongst other causes in the treatment and composition of ores which render gold un-amalgamable in the mill, and which have to be considered and guarded against, Dr. Egleston mentions :—

1. Gold which has been hammered, so as to increase its density and close its pores, amalgamates very slowly, a condition, however, which can scarcely be taken into account in the working of a stamp-battery, unless the gold is very coarse.
2. The presence of sulphuretted hydrogen in the water, likely to be induced by the presence of soluble sulphides.
3. Gold which has been exposed to the vapour of sulphur, which no doubt accounts for the behaviour of certain ores when submitted to amalgamation after roasting, and the same is doubtless true of arsenic in some instances, judging from Mr. Pearce's conclusions.
4. Certain alloys of gold, like the phosphide, though others like the arsenide and antimonide (produced artificially) are apparently more tractable. To these alloys must now be added the bismuthide, telluride, and cupride.
5. Greasy substances, like powdered natural hydrated silicates of magnesia and alumina, if present in the ore, froth and coat the gold with a slime, preventing the action of the mercury, and lubricants such as oil, getting into the battery box, are fatal to gold amalgamation.

The late Mr. E. N. Riotte once mentioned in the writer's presence a case within his experience, which occurred at a gold mine in one of the Southern States, where they had to abandon the use of dynamite on this account; as owing to the inexperience of the negroes in its use, portions of unexploded cartridges were left in the stone, and affected the battery yield. The result, if the nitro-glycerine had leached out, and collected in a corner of the box, would doubtless have been a revelation to them.

6. Unusually soft water, absorbing air or carbonic acid gas, and so tending to form ferric sulphate (when pyrites is present), which is a most active agent in staining the plates.

Glancing back at the processes that have been referred to, it will be seen, that except in dealing with free-milling silver ores and a few special cases, there is considerable latitude given in the choice of a process, not to mention its details.

Its proper selection will depend on considerations of relative cost, as compared with saving or loss of metal, or other valuable products, that the ore contains; and nothing influences both of these in the same way, or either of them more, than the degree to which the crushing of the stone is carried, which is the author's next proposition.

The finer the crushing, and the more the ore is handled or re-handled, by repeating the operation, the greater of course will be the expense; while the more it is handled and the finer it is crushed beyond a point that is absolutely necessary, the greater also will be the loss. To reduce this latter to a minimum, a certain cost must naturally be incurred, though if it go a little too far, the cost is increased and the loss as well.

This happens under various circumstances, viz. :—

A. In coarse concentration of the baser metals and of gold and silver ores, or a combination of both (applicable to the latter classes of ore, when the sulphides exceed say 10 to 20 per cent. by weight) to obtain the mineral with a minimum loss from comminution, the ore must be broken up, only just enough to unlock all the mineral it may contain, down to a certain size, and (theoretically) the mineral and barren rock with it should be separated at once. This will leave, however, a certain amount of yet more finely divided mineral in the residues, which require to be re-crushed and re-treated at so much additional cost, to get out the metal that remains in them, less of course a certain percentage of loss involved in this re-treatment.

When coarse combined with fine-concentration is necessary, it may then be considered how far it will pay, to carry theory into practice, and when labour is cheap, the life of the mine assured for a long period, and the ore of considerable value, the German practice may be adopted, of classifying carefully, reducing the ore gradually, and concentrating close; but when labour is dear, the life of the mine uncertain, and the chief object is to secure the largest possible profit in the shortest possible time at a minimum capital outlay, irrespective of a certain unavoidable loss in unrecovered mineral, the Anglo-American method must be adopted of sacrificing saving of mineral, to saving of cost and time, and crush the ore in quantity, to a much greater average degree of fineness, without such careful regard to sizing it.

The latter practice is the one followed in most American and colonial mills, employing fine concentration for gold and silver ores (when the sulphides in the ore do not run over, say, 10 per cent.). To compare

these divergent systems roughly as to cost, two different cases, one of a European zinc-and-silver lead-mine, the other an American lead-mine may be taken.

The works in Europe are those of the New Pierrefitte Co., which the writer was commissioned to inspect last summer, and of which he is permitted to give the following particulars :—

They are situated in the Pyrenees in France, and treat an ore containing galena, blende, magnetic iron, and a small amount of copper pyrites.

On an output of 21,449 tons of crude ore last year (averaging 9·33 per cent of silver-lead, and 11 per cent. of zinc) the cost of dressing, covering wages, supplies (including coal for three months of the year, water-power being used for the remainder), renewals of machinery, etc., may be taken at about 3s. 10d. per crude ton, as given below :—

Continental Works. Per Crude Ton. Francs.		American Works. Per Ton. Cents.
Labour	2·35	Labour ... ... 13·4
Supplies and repairs ...	2·43	Supplies ... ... 3·5
		Repairs ... ... 10·0
		Coal ... ... 9·5
Total ...	4·78 = 3s. 10d.	Total ... 36·4 = 1s. 6¼d.

The American case is that of the new dressing works of the St. Joseph Lead Co., at Bonne Terre, Missouri.* The ore is non-argentiferous galena, associated with some iron pyrites, carrying traces of nickel and cobalt. On an output of 224,203 tons of crude ore in 1886-1887, averaging 5·65 per cent. of galena, the cost of dressing for the fiscal year, ending May 1st, covering labour, supplies, repairs, and coal, was (1s. 6¼d.) 36·4 cents, divided as above shown.

It would take too long to explain the differences of treatment in the two mills, suffice it to say, that the system outlined in both, is to crush with rock-breakers and rolls (preceded at Pierrefitte by spalling and cobbing), and then to jig and wash the ore on tables and buddles. The difference mainly is, that in the one case they give attention to sizing, and in the other they do not, though each mill in detail possesses certain peculiarities of its own. Neither can the relative number of workmen, cost of wages, supplies, etc., in detail be compared, as these data in the cost sheet are wanting, in the otherwise excellent paper quoted, from which the particulars of the Missouri mill are taken. In the French works, wages range from 1 fr. (10d.) to 2 fr. (1s. 8d.) for boys, and 2·75 fr. (2s. 2½d.) to 3 fr. (2s. 6d.) per day for the larger number of adult

**Trans. Am. Inst. Min. E.*, vol. xvii., page 659, H. S. Munroe.

mill hands, and it is noteworthy that in both mills repairs and supplies (excluding coal from the American case) bear nearly the same ratio to one another.

It would appear at first sight, that all the advantages lay with the American mill, and no doubt it does in America, where labour is dear, with the base ores that have to be dealt with in quantity, where the works are situated. The case, however, would be entirely different if the American mill were transplanted to the Pyrenees, and set to work the silver ores of that locality, as the loss in the tailings at Bonne Terre is about 2·13 per cent., or 27·4 per cent. of the total amount of lead in the ore. These large losses are due to included mineral in the coarse-sands, and to very finely divided mineral in the very fine slimes, as a large part of the ore requires very fine crushing, owing to the exceedingly minute state of division of a large part of the mineral.

In the Pierrefitte works such losses with argentiferous galena could not be afforded, and though the writer is not in a position to state what they are exactly, they are far less than in Missouri. Moreover, the cost in the Pyrenees, with a larger output from the mine, could be materially reduced, as the works are not run up to their full capacity, and if their general design could be remodelled on the lines of more modern Continental mill-construction, the expense of treatment (if carried out on a similar scale as regards quantity) might, be made to compare more favourably with the price in America without sacrificing mineral, in view of the lower price of labour.

To again quote Prof. Munroe, the losses in the Lake Superior copper mills range, it is said, from 28·5 to 31 per cent., treating mineral much more easily saved than the Bonne Terre galena, whilst the cost of dressing copper at the Atlantic mill in 1885, using steam stamps, is stated to have reached 30·36 cents (1s. 3½d.), divided between labour, 5¾d.; fuel, 7¼d.; supplies, 2½d. These costs were reduced, however, at the Atlantic mine in 1887 to 27·5 cents or 1s. 1¾d. per ton of copper ore treated. Incidentally, Prof. Munroe states it is cheaper to crush ore* with rolls, arranged as they are in the Bonne Terre works, than the Lake Superior amygdaloid with steam stamps, but this is partly due to the friability of the Missouri ore; what is far more remarkable, however, is that the rolls produce (according to the Professor's statement) quite as large a proportion of slimes as the steam stamps.

Further, it should be noticed that the above instances of the cost of concentration in America are to be looked upon as a good deal below the

* Limestone.

average (at any rate of smaller mills), and as a general rule, coarse concentration, where rolls are employed for crushing, does not cost far short of 75 cents or 3s. 1½d. per ton, where labour rates are dear.*

The average cost of dressing different classes of ore in France is given by Dr. Foster and Mr. Galloway† (from a statement of Mr. Huet) as:—

	Per Ton. s.	d.
Iron ore	0	2
Coarse-grained galena	5	10
Fine-grained galena	7	10
Manganese ore	7	10
Copper pyrites, or grey copper ore with iron pyrites	9	9½
Coarse galena and blende	9	10
Fine galena and blende	11	10
Copper pyrites, or grey copper ore with galena ...	13	10
Copper ore, galena and blende	20	4

Though these figures may have been typical and true of French practice at the time they were compiled, the dressing-costs at Pierrefitte would indicate that they are much above the average at the present time; and, as the translators remark, there is always great uncertainty in estimating such working costs in a general way, for there is always uncertainty with reference to the labour required for shifting the stuff about, and principally with reference to the loss in dressing, to which might be added the attention required by different classes of machines when differently grouped.

If the American losses in dressing be compared with those in Germany it will be seen that the advantage rests with the Continental system.

Oberbergrath O. Bilharz, in a paper entitled "Die Neue Central Aufbereitungs-Werkstätte der Grube Himmelfahrt, bei Freiberg i.S.," puts the losses in the tailings of these works, at only 0·01 per cent. silver, *nil* per cent. lead, 10 per cent. sulphur, and 9 per cent. zinc. While the settlings in the final catch-pits‡ carry only 0·01 per cent. silver, 2 per cent. lead, 8 per cent. sulphur, and 6 per cent. zinc.

Quoting the pamphlet referred to, the transcript of an article from *Industries* (in the *New York Mining Journal* of February 20th, 1892) and the writer's personal knowledge, he would like to refer to the Freiberg works more particularly, because he thinks they are illustrative of American principles, so to speak, engrafted upon former German practice,

* In exceptional cases the price may run up to 12s. 6d.

† *Lectures on Mining*, by J. Callon, vol. iii., page 134. Translated by C. Le Neve Foster, D.Sc., and W. Galloway.

‡ Only a fractional portion of the crude ore, which is re-treated.

presenting a model of economy in costs, as well as economy in saving of mineral, to an extent which has never been achieved before, an advance which is undoubtedly in the right direction.

The ores which are obtained from the various shafts of the Himmelfahrt mine consist of argentiferous galena and zinc blende, with iron, copper, and arsenical pyrites, while the gangue is partly quartzose, partly sparry, mixed with the country (gneiss).

On account of the variety of the ores of the Freiberg district, the machinery of the dressing-floors is duplicated, so that ore from other mines can be dressed apart in the two sections, into which the mill is divided.

The annual production of the Himmelfahrt mines alone is 45.000 crude tons, of which four-fifths is concentrating ore, coming from what is known as the pyritic-lead formation (kiesige Blei-Formation) the remainder being rich silver ore and high-grade hand-picked galena ore, found mostly in cross-courses, carrying a gangue of brown spar and heavy spar.

These latter ores, representing one-fifth of the total production, are separated dry (hand-sorted and dry-stamped) and sent direct to the smelting-works.

The dressing-floors* are designed to handle 150 tons of pyritic ore per shift of ten hours, each separate section dealing with 75 tons, with an average total consumption of 35 cubic feet of water per minute, the machinery being driven by steam-power.

The ore averages 0·15 to 0·20 per cent. silver; and iron pyrites and galena are the predominating minerals. The zinc blende is black, and contains 33 per cent. of iron. The building is arranged in storeys, one above the other, and contains rock-breakers, screens, hand-picking tables, several sets of rolls, wet gravitation-stamps, a variety of revolving screens and jigs, hydraulic classifiers, Stein vanning-frames, and other machinery (fully described in the papers referred to) which present many novel features of detail.

The horizontal compound-condensing engine which runs the whole crushing, dressing, and electric-lighting plant, indicates 105 horse-power, and the works employ 44 workmen, with 3 overseers, 1 engine-driver, 1 stoker, and 5 fitters, in all 54 men.

The cost of dressing one ton of ore amounts to 10d.

* The machinery was built and erected by Mr. C. Luhrig, of the firm of Messrs. Luhrig, of Dresden, in accordance with the plans of Mr. O. Bilharz, who designed the installation.

There are only, five points which could be charged against the plant taken as a whole :—

1. That the design involves a considerable capital outlay (an objection, which it will be shown later on, is entirely swept away, when it conduces to and results in economy of treatment, provided that the mine warrants it, and that the working capital of a company can stand the necessary call without stinting the development of the mine.)*
2. That it may be doubted whether efficient roller-mills of the Schranz type, might not advantageously be substituted for the stamps.
3. That, large as may be the reduction in labour as compared with many other Continental plants of the same capacity, it might be still further lessened, given a better site than the one at Freiberg, which possesses but a moderate fall.†
4. That owing to the nature of the machinery, several skilled machinists, as well as facilities for making repairs on the spot, are very essential for running it properly.
5. That if anything goes wrong with any of its individual parts, the whole plant appears to be brought to a standstill. The writer was reminded of this by the practical illustration of a belt slipping off a pulley whilst he was going through the works two years ago. It is a matter, however, which is capable of remedy.

On the whole the advantages far outweigh any objections that can be urged, and the structural details of the plant, and erection of the machinery exhibit first-class skilled workmanship.

The stock required to be kept on hand in concentration-works consists of duplicate parts of the machines, in more or less number, (depending on the distance from a source of supply and the average life of the different pieces in use).

The proposition stated on page 23 holds good.

B. In the fine concentration of sulphides (which generally accompany the precious metals) on belt or other automatic concentrators, which are so frequently used‡ as an adjunct to the stamp-battery.

* As these works are a Government concern, this last consideration may in the present case be dismissed.

† Owing no doubt to its being the only site available on account of other reasons.

‡ Before adding concentrators of this kind to a mill, it is of course necessary to ascertain that the clean concentrates will assay enough to leave a profit on further local treatment, or shipment, to extract the gold, silver, or other metals they may contain.

The size to which the stone is reduced under such circumstances influences very largely the chance of loss of mineral and gold, whether included in the coarser part of the tailings, or liberated as fine slime.

The cost of the actual mechanical fine concentration is not a matter of extremely serious consideration in itself, since assuming 18 tons of concentrates saved *per diem* in this way, crushing an ore containing 2½ per cent. of pyrites, the cost of treatment would not exceed 2d. or 3d. per ton, and dealing with much smaller quantities, 6d.; and further it will not vary much whether the ore is stamped coarse or fine, unless it involves a double concentration (that has been alluded to) dealing with rich ores liable to slime. But in order to concentrate in this way the ore must be stamped wet, and what applies to crushing fine with rolls, is also true of stamps. The finer the crushing the more it must cost in time, representing extra labour and a wear-and-tear, and also fuel, etc.

C. In stamping previous to grinding in pans, if carried too far, there will again be loss in the battery-tails, whilst if not carried far enough, it will give the pans unnecessary work to perform, which would be much more cheaply done in the battery.

D. When the concentrates are treated (obtained by coarse or fine concentration or both combined) by such processes as smelting, chlorination, and lixiviation, it will be found that, if it involves a preliminary roasting (as these processes generally do), if the ore is crushed very fine, and it carries much lead for instance, it entails a more careful regulation of the temperature, and unless a suitable type of furnace is used, with extensive dust chambers attached, great losses both mechanically and by volatilization may be incurred. On the other hand, if the ore is not crushed fine enough, and it is subjected to a chloridizing roasting with salt, the chlorination will not be as high as it should be, and the subsequent treatment will be proportionately affected by it.

E. If the ore is to be lixiviated or chlorinated, and the stone is crushed too fine, and at the same time of too uniform a grade, great loss of time and imperfect filtration and leaching will result, more especially if it is at all of a clayey nature.

The choice of a process, whether for treating crude ore or concentrates, must be governed by the relative commercial profits it promises, as compared with other methods of treatment. But before attempting to touch

this question, it will be advisable to state certain general facts (under separate headings), giving an outline of several of the processes, which are later on compared, under the head of rival methods of ore-treatment, in order to simplify the subject.

Roasting and Chlorination.

This is, ordinarily the best process to employ, for the treatment of the high-grade pyritic concentrates of stamp-mills, and hand-picked sulphide ores, provided a sufficient quantity can be commanded to run the works full-time or nearly so; intermittent work being bad for plant as well as for men. The ore should be kept moist after concentration, till it goes to the drying-floors, as otherwise lumps will form which will not roast properly unless re-crushed.

The ore to be roasted is generally worked in reverberatory furnaces to expel the sulphur, arsenic, and other volatile compounds till it is dead sweet or as nearly so as possible. Then a small quantity of common salt is added, and the silver is chloridized, the sulphur is all driven off, and the ferrous and cuprous sulphates are oxidized.

The ore having been damped to about 6 per cent. is gassed with chlorine, a trichloride of gold being formed, which is leached out and precipitated with a solution of sulphate of iron. This gold is collected on filters, thoroughly washed, dried, and melted, and should average from 998 to 999½ fine.

When silver is present in sufficient quantity to justify the extra treatment, it can be recovered by re-leaching the residues with hyposulphite of soda or lime to dissolve the chloride of silver, which is precipitated from the lixivium by the addition of a solution of polysulphide of sodium or calcium, and the sulphide of silver resulting (collected on the filters) is washed, dried, and reduced to the metallic state.

There is, however, one point to be guarded against in adopting this latter mode of treatment in dealing with ores, in which the percentage of gold is high, and about equal to the silver.

Kustel,* remarks:—"If such an ore should be subjected to chloridizing roasting, then impregnated with chlorine gas, leached with water for the purpose of extracting the gold, and finally leached with hyposulphite of lime for precipitating the silver, it would in this case, although a high percentage of silver might be extracted, result in a yield of gold that would hardly amount to 50 per cent., more or less.

* *Roasting of Gold and Silver Ores*, by G. Kustel, second edition, page 123.

The reason is not easily explained. The gold may be influenced somehow by the base-metal chlorides during the roasting, which prevents the gold being attacked by the chlorine gas."

On the other hand, if the base-metal chlorides and the chlorides of silver are extracted previous to the impregnation with chlorine, both metals, silver and gold, can be got out very close by a process invented and patented by O. Hofmann, which Kustel fully describes in detail.

The gold and silver-bearing sulphides of the Colorado No. 2, G. and S. M. Co., at Monitor, Alpine County, California (as an instance) were successfully treated by this method.

Should the sulphides contain any lead, it is advisable to conduct the liquor from the leaching-vats to settling or storage-tanks, and about 40 lbs. of sulphuric acid (66 degs. B) is added. By this addition the lead is precipitated as sulphate, and the liquor, being freed from lead, can yield no plumbic sulphate with the gold, as it would otherwise do, when precipitated with sulphate of iron; hence a cleaner bullion.

In some chlorination works,* as the sulphate of lead obtained from the base tanks, always contains some gold, it is collected and sold, and a considerable sum is then realized for both lead and gold. The wooden tanks are protected from the action of the acids, by a coating of paraffin paint; and the covers of the chlorination-vats are luted on with a mixture of tailings, bran and water.

There are certain pyritic ores, however, to which chlorination is unsuited. These include amongst others :—

1. Ores of low grade, which run below, say £3 to £4 value per ton.
2. Those in which the gold is very coarse.
3. Those in which there is any mineral present, or other metal except silver and gold, in the gangue, liable to be attacked and rendered soluble by the chlorine, whilst gold must be either in a metallic state, or in a combination which can be destroyed by roasting without loss.

The impurities most to be avoided are sulphur, antimony, and arsenic, since soluble salts of the base-metals may precipitate the gold in the leaching-vat; hence the necessity of what is termed a dead sweet roast before-mentioned. Lead, lime, and magnesia are also deleterious, as they are attacked by the chlorine, and waste a great deal of it, besides introducing difficulties in the after-precipitation.

Mr. Nelson E. Ferry, M.E., in the *New York Engineering and Mining Journal* of November 28th, 1885, recommended the addition of molasses

* *Eighth Report of the California State Mineralogist*, page 47.

to the leach when lime was present. Mr. Ferry says: Dissolve one gallon of molasses in 30 or 40 gallons of water and keep for use. The quantity to use in each case must be determined by a laboratory test. If calcium sulphate comes down, either the molasses is in insufficient quantity, or it has not been thoroughly mixed. Examine by transmitted light. Avoid large excess of ferrous sulphate. If the gold comes down at first in a flocculent state, that does not matter, it soon assumes the usual form. The best results are got when the liquid is made slightly acid.

Mr. A. H. Aaron* recommends the use of precipitated copper sulphide as a precipitant,† stirring it into the gold solution, or better, allowing the gold solution to flow through a series of small filters containing the sulphides. He states as a reason that, unless there is copper present to precipitate it, there is often a considerable loss from gold in suspension, due to imperfect settling.

The presence of galena, necessitates a good roasting with a strong finishing heat, as far as its fusibility will allow. This involves the use of a long hearth to raise the temperature gradually, as the charge progresses from one end of the furnace to the other. The roasted ore must be examined to see that no galena is left undecomposed.

Any soluble iron chloride or other soluble metallic salt formed, will react on the iron oxide and precipitate the gold, when dissolved, leaving it in the tails of the ore-tanks after lixiviation. The addition of a little salt towards the close of the roasting, after the ore has been carefully oxidized, tends to counteract the effect of the lime and magnesia so often present in gold and silver-bearing rocks.

Concentrates frequently contain iron as well as copper pyrites, galena,

* *Notes on the Hydro-metallurgy of Gold.*

† Mr. Claude Vautin, whilst admitting that precipitated copper sulphide (CuS) is a good precipitant in the laboratory, points out that in consequence of its physical condition, and the facility with which it is oxidized to $CuSO_4$, its application in practice is not to be recommended. He recommends fused sub-sulphide of copper (Cu_2S) as a perfect and rapid means of recovering the gold. To facilitate the filtration the Cu_2S is crushed to pass a 60 mesh sieve, and the portion which remains on a 100 mesh sieve only is used; this gives the desiderata of a dry, hard, and granular filter-bed, three points of importance. Whilst charcoal is not free from objections, on account of the large quantity necessary to ensure a complete decomposition of the solution, and the somewhat tedious and troublesome operation of "burning-off" the excess of carbon before smelting the residues, Mr. Vautin claims for this method the advantage that it is not necessary to expel by heat or otherwise any excess of free chlorine from the solution before passing it through the reagent; nor does the presence of any free hydrochloric acid in any way interfere with the reaction. Any copper, calcium, zinc, etc., also passes through, presenting advantages over the use of hydrogen sulphide or ferrous sulphate.

and arsenical pyrites, combinations which do not prevent the application of the process, unless the base-metals, more particularly lead (but always excepting iron), are present in large proportions. Losses are likely to occur through the gold being volatilized or affected in roasting, which is more particularly to be guarded against if copper is present, and the ore be roasted with common salt. Much has been written on this subject, and in fact, enormous losses may occur in the chloridizing roasting of gold ores through volatilization of the gold.*

Mr. Kustel, page 57, records the loss of 20 per cent. of the gold contents in the oxidizing-roasting of certain tellurides of gold and silver, and states it is not a mechanical loss, but is due to volatilization. Though with most ores no loss of gold is suffered in this way during oxidizing-roasting either with iron or arsenical pyrites, loss may occur if the operation be carried on so rapidly that fine particles are carried off by the draught.

A loss of silver in oxidizing-roasting is unavoidable. Plattner† concludes that the percentage increases with the temperature of roasting, and with the looseness or porosity of the roasting charge, that is with the facility with which the air can come into contact with the silver, and the freedom of the silver from combination with other substances. The loss increases also with the time of roasting. He concludes that silver is volatilized as oxide, which decomposes at a lower temperature into silver and oxygen, but Dr. Percy throws doubt on this theory.

In roasting, it is easy to incur an enormous loss of gold by inattention to what may appear insignificant trifles, showing how necessary in such matters is a systematic weighing and sampling, as well as assaying of the ores and products, in order to know at once when such losses are taking place, so as to be able to check them in time. Haphazard and occasional sampling and assaying, are worse than useless; they lead to great losses of valuable capital, frequently to the total abandonment of good properties, and worse than all, to a false sense of self-satisfaction, which discourages investigation and improvement by denying the necessity thereof.

If the gold in the ore is of low fineness or combined with silver, a large part of it may be lost in roasting from a different cause, as the subchloride will surround the particles of gold, and prevent any further action of the chlorine upon it, in which case the double process, previously alluded to, must be employed.

The quantity of common salt used in chloridizing-roasting, and the time when it can best be added, are of much importance to the final result;

* "Losses in Roasting Gold Ores and the Volatility of Gold." *Trans. Am. Inst. Min. E.*, vol. xvii., page 3.

† *Metallurgische Rostprozesse*, Freib, 1856.

and they have to be determined by trial, with different ores. The chlorine must be purified from everything that would be likely to cause a reaction between it and the other constituents of the ore.

Other important factors in the general success of the process are :—

1. Close and clean concentration.
2. Amalgamation of any coarse gold, preceding or following the chlorination.
3. Lixiviation of the silver before or after gassing.
4. Absence of any organic matter in the charge, or wash-water.
5. That the ore is crushed properly. A series of experiments should be made to ascertain how coarse the ore should be to give the best results with reference to economy, large capacity, and best extraction. The pulp for the best leaching must be in a granular condition, and carry as small a percentage of dust and slimes as possible. Mr. John E. Rothwell (*New York Mining Journal* of February 7th, 1891) points out that for this purpose rolls, properly managed, are best adapted for this purpose. The chief point is to make the reduction in the size of the particles passed through them gradual. The ore should come to the coarse rolls not larger than $\frac{3}{4}$ inch, and these rolls should crush to about $\frac{3}{8}$ inch. The middle rolls are set about $\frac{3}{16}$ inch or less apart, and the fine rolls about as far apart as the size to which the ore must be crushed. If only two sets of rolls are used, the coarse are set a little closer, and the fine remain the same. The springs should be tight enough in tension, not to give with the hardest ore passing through them, but lax enough to allow a piece of steel or iron to pass through without throwing off the belts. The periphery speed of the rolls should be the same or a little faster than the falling speed of the ore, and the ore should be fed in an even sheet across the surface of the roll. This will keep the surfaces true and produce a granular pulp but carrying a small percentage of dust. A still more gradual reduction can be made by making the rolls of larger diameter and narrower, which will give them also proportionately a greater capacity. Rolls of 39·5 inches diameter and 12 to 15 inches face have been used by Mr. Rothwell with good results.

Dr. T. Egleston writing on the formation of gold nuggets and placer deposits says :—"I have known of gold (Grass Valley, California) being

thrown down on the filter of a Plattner vat by the organic matter contained in the very impure wash-water used for the solution of the gold, rendered soluble by the action of the chlorine. The filter was full of metallic gold, and there was no means of ascertaining how much had been lost. Several ounces of a brown deposit were taken from it, almost pure gold."

An excess of chlorine must be present in the generator, and the ore must be washed and filtered thoroughly, otherwise the tub tails will assay 4 to 5 dollars instead of 75 cents.*

LIXIVIATION.

The MacArthur-Forrest Process for Gold Ores.

This process claims† to extract the gold from pyritic ores, without the necessity of roasting; and in certain cases it may have a considerable field of usefulness. When, for instance, the gold is in an extremely fine state of division, and the ore contains silver as well as gold, the fairly high percentage extracted of each metal may render this method a very desirable one.

For plain gold ores, in which the gold is in fine particles, the barrel-chlorination process seems, however, to give a higher extraction, and with ores of moderate and high-grade appears to work to better advantage.

It would appear, in fact, that the limitations of the cyanide process might be summed up as follows:—

1. That it is only entirely successful with free-milling ores, as although it will deal with pyritic ore, it does so at a greatly enhanced cost.
2. That it is inapplicable to ores containing a considerable percentage of coarse gold.
3. That it cannot be economically applied to rich material, as a loss of 2 dwts. in 8 dwts. ore is a very different matter to a loss of say 2 ozs. in 8 ozs. ore.
4. That it is not applicable to ores containing certain metal and mineral combinations.

On the other hand, it appears to be a process admirably adapted for saving gold in the condition of float, *i.e.* in an extremely minute state of subdivision, which cannot be caught by any process of concentration or amalgamation.

* Phillips, *Trans. Am. Inst. Min. E.*, vol. xvii., page 313.

† *Memoranda on the Treatment of Refractory Gold Ores by the MacArthur-Forrest Process.* Wm. Hodge & Co., Glasgow.

Briefly outlined, the MacArthur-Forrest process originally consisted in pulverizing the ore to 40 or 60 mesh size, and then mixing, and agitating it, with a solution of cyanide of potassium (the ordinary standard strength of which was intended to be $1\frac{1}{2}$ parts of cyanide to 100 parts of water by weight).*

After a sufficient time has elapsed for the solution of the precious metals, the leach is transferred to large wooden filter-tanks, the solutions are allowed to settle, and are drawn off (sometimes assisted by pressure or suction), and the gold and silver contained in these solutions as cyanides, are decomposed and precipitated in a metallic condition, by passing the filtrate through metallic zinc turnings. In the course of this treatment the zinc replaces the gold in solution as cyanide of zinc, which dissolves in the water, while the gold is deposited as a dark powder. This is separated by sieving from the undecomposed zinc, when it is found, by testing with chloride of tin, that the whole of the gold has been precipitated from the leach liquor.

The zinc residues are then removed and dissolved in nitric acid, the gold remaining undissolved as a dark brown powder, which is washed, dried, and melted, yielding almost pure gold.

If there is any silver present it is dissolved with the zinc, and can be recovered by adding a solution of common salt, which throws it down as chloride. It has then to be reduced to the metallic state by contact with sheet-iron or zinc, after which it is washed, dried, and melted.

The chemistry of the process† is simple, depending on the affinity of cyanogen for gold and silver, and the ease with which these metals form soluble double cyanides with the alkali metals. The relative affinities of the different metals, according to Mr. Wm. Jones, stand as follows:— 1st gold, 2nd silver, 3rd copper, 4th zinc, lead, arsenic, antimony, etc.

The solvent action on the base-metals can be reduced to a minimum by reducing the strength of the solutions, the readily soluble gold and silver being dissolved out with only traces of copper, zinc, etc.

The best strengths of solutions to use in leaching out the gold from refractory stone depends entirely on the nature of each ore, and it is

* In actual practice, however, these ideas have been modified, and what is known as the percolation system has come into almost universal use, employing standard solutions of a certain strength, first leaching the ore for 6 to 12 hours with .6 to .8 per cent solution, and for 8 to 10 hours more with a weaker liquor, containing .2 to .4 per cent. of cyanide.

† "The MacArthur-Forrest Process for the Treatment of Refractory Gold Ores." *Eng. and Min. Jour.*, New York, 1889, vol. xlviii., page 544.

impossible to lay down any hard-and-fast line. The point must be determined by practical tests.

Filtration of the liquor is accelerated by using a vacuum, and there is no practical difficulty about this, unless there is a large percentage of clayey matter present. The amount of free cyanide in the liquors, after passing through the zinc, is then determined by means of a standard solution of nitrate of silver, and the liquor is thereafter made up to its original strength and used over again.

To extract the gold from refractory ores a number of points must however be observed. If the ore betrays a noted acidity, due to the presence of basic sulphates of iron, etc. (especially marked in the case of disintegrated and weathered sulphides), it should be neutralized with an equivalent quantity of caustic lime in the form of milk of lime. The exact amount of acidity can be determined by shaking up a weighed sample of the ore with water, and adding standard-normal or tenth normal caustic-soda solution till the point of alkalinity is attained, as indicated by litmus or any other suitable tests. The amount of lime is then easily calculated. Some ores show as much as 4 per cent. of acidity in terms of soda, and such ores, on treatment with cyanide solutions without previous treatment with lime, fail to yield their gold contents, whereas when previously treated with lime, the greater part of the gold is easily extracted.

Nearly all sulphides show more or less acidity, but when it is under 0·10 per cent. it may for practical purposes be neglected.

The cyanide solution used should be as free from caustic alkali (NaHO or KHO) as possible, as it is apt to form a sulphide of sodium or potassium with the sulphur of the ores, and thus prevent the gold and silver going into solution. This difficulty, when it does occur, is got over by adding calcium chloride.

The cyanide solutions are best preserved from too great exposure to the air, as a part of the cyanide is apt to be converted by oxidation into the cyanate.

From a chemical point of view it appears, that the economic success of the process, will mainly turn on the price, and consumption of chemicals, and on the time taken in treating large quantities, to extract a given percentage of gold from various combinations of ores in different localities in which the gold exists in a coarse or fine state of division, and in different alloyage, and in which the pyrites is more or less highly concentrated. The ores of the Witwatersrandt seem specially adapted for treatment by this process.

The Russel Process for Silver Ores.

Quoting Mr. C. A. Stetefeldt, on the lixiviation of silver ores, Mr. E. H. Russel, it is said, discovered that a solution of a double salt of cuprous hyposulphite and sodium hyposulphite (formed by mixing sodium hyposulphite with copper sulphate) exert a most energetic dissolving and decomposing action upon metallic silver, silver sulphide, silver minerals belonging to the group of antimonial and arsenical sulphides, and other silver combinations. Hence, if a charge of roasted ore is first lixiviated with ordinary sodium hyposulphite solution to dissolve the silver chloride, and subsequently with cuprous hyposulphite (this solvent is called the extra solution) an additional amount of silver is extracted, which would have been lost in the tailings, by working according to the old method alone. Or, if the roasted ore contain caustic lime and be treated with the extra solution, the deleterious influence of the caustic lime is thereby neutralized. In the same way the extra solution may be applied to extract silver from raw ores without previous chloridizing-roasting, or to lixiviate ores after they have been subjected to an oxidizing-roasting. Mr. Russel also discovered that lead can be completely separated from a sodium hyposulphite solution, as lead carbonate, by sodium carbonate, without precipitating copper or silver.

After decanting the solution from the lead carbonate, silver and copper are obtained from it in the usual way. This method of separating lead prohibits the use of calcium polysulphide as a precipitant for the sulphides, because calcium, entering the regenerated lixiviation-solutions, would also be precipitated as a carbonate with the lead, by sodium carbonate; hence sodium sulphide must be employed. A full investigation has demonstrated that this is by no means detrimental, as sodium sulphide and sodium hyposulphite are more advantageously used in the lixiviation process than the corresponding calcium salts.

Finally, Mr. Russel found that if a hyposulphite solution has a caustic reaction, produced by caustic soda or lime, its solvent power for silver is materially deteriorated. This defect he corrects by neutralizing such a solution with sulphuric acid.

The Van Patera Process (Ordinary Lixiviation).

This process consists in:—(1) Crushing the ore (generally with rolls). (2) Drying the ore. (3) Roasting with salt. (4) Leaching out the base-metals with hot or cold water.* (5) Leaching out the silver with

* The base-metal chlorides in the water may carry off 0·5 to 3 per cent. of silver, but this can be recovered by simple dilution, unless chlorides of lead and antimony be present (which are likewise precipitated), in which case either the ore must be leached from below, under slight pressure, or the silver must be precipitated in troughs outside the tubs.

hyposulphite of soda. (6) Precipitating the silver. (7) Roasting the sulphide of silver and melting to bullion.

It is capable of saving about 85 per cent. of the silver in the chloridized ore, assuming it to be suited to this class of treatment.

The writer will not stop to refer to other wet methods, such as the Claudet and the Kiss, for gold and silver ores, and the Augustine and Ziervogel, used for the treatment of argentiferous copper matte, though they all undoubtedly possess special features of interest.

The difference between the Russel process and ordinary lixiviation, must however be alluded to.

1. It requires a less careful chloridizing-roasting, and on that account a lower percentage of common salt may be used in roasting.
2. It extracts a higher percentage of silver by means of the extra solution. This is specially of importance in treating raw ores, and in lixiviating roasted ores containing caustic lime.
3. It produces sulphides free from lead.
4. It yields lead in the form of lead carbonate as a valuable bye-product.
5. It overcomes the deleterious effect of a caustic lixiviation-solution, by neutralizing it with sulphuric acid.
6. It uses sodium hyposulphite and sulphide exclusively, and not the corresponding calcium salts.

Ores suitable for Treatment by the Russel Process.—It is claimed by Mr. Stetefeldt that almost all silver ores that do not carry a large percentage of lead or copper, can be treated by lixiviation with success and economy. He says :—"I do not mean to create the impression that, from ores containing more or less lead and copper, a high percentage of the silver cannot be extracted by this process. Such ores, however, will in most localities be reduced to better advantage by smelting."

It is safe to state that all ores fit for amalgamation can also be treated by lixiviation, and that the Russel process may succeed where amalgamation is a failure. In cases where lead-bearing silver ores are suitable for concentration, it may be profitable to concentrate the ore by the Krom dry system, obtaining and smelting product, high in lead, and to lixiviate after roasting the tailings and the dust.

The dry system of concentration deserves the preference, because it delivers the tailings and the dust in a condition ready for chloridizing-roasting. In wet concentration, the drying of the tailings would be expensive, and there would be a considerable loss of silver in slimes.

Oxidized ores, containing silver chloride and lead minerals, may be lixiviated after crushing, and the tailings may be concentrated for lead. In this case wet concentration would be most suitable. This has been done at the Old Telegraph mine, Utah. The Russel process is also adapted to the treatment of tailings, resulting from ores which have been worked either by the old lixiviation process, or by amalgamation. Whether it is more profitable to lixiviate an ore raw, or after chloridizing-roasting, or after oxidizing-roasting, must be determined in each case by actual experiment.

Personally, on commercial and other grounds the writer cannot take Mr. Stetefeldt's view with regard to using dry concentration as an adjunct to lixiviation, unless recent improvements have been made in the process, which he is unacquainted with; though, of course, this is merely a side question, and on the main issue Mr. Stetefeldt is no doubt one of our greatest living authorities.

Smelting.

Although this is one of the most important methods of beneficiating ores of gold and silver, such as contain large amounts of lead and copper (a class by themselves), which can rarely be treated in any other way, the writer does not propose going into this branch of the subject, beyond stating that favourable local conditions with regard to nature and grade of the ore, and the possession of cheap and suitable fuel and transport, skilled labour, fluxes, dump-accommodation, water, and other things are necessary to render works of the kind a success. A convenient central situation, and regular supply of ore in sufficient quantity, to keep the furnaces running steadily, being two most important points.

Notwithstanding, however, that the gross cost of a smelting-process is large, the high value of the products it yields frequently justifies its employment, even in comparatively out-of-the-way parts of the world.

Under circumstances of this kind water-jacketed furnaces of the Piltz type, seem to be gaining more and more ground in argentiferous-lead smelting in the Western States of America, and are most highly successful, more particularly as regards cost, which precludes the use of high-class metallurgical fuel (in many places practically unprocurable).

Every metallurgical process gives rise to a certain amount of loss.

The losses in roasting have been already referred to, and some of their causes explained, and as roasting is necessary to prepare most classes of ore* for smelting, it is well to bear the facts in mind.

* If they contain an excess of 5 per cent. of sulphur.

A further loss ensues in the actual smelting, part of the precious and base metals being volatilized or mechanically carried out of the furnace, and parts entangled in the slag, while a still further loss takes place in refining the lead bullion.

As a rule, the loss in smelting proper in the Western States of America, may be put down, the writer believes, at 3 to 4 per cent. of the silver, and a variable amount of lead, depending chiefly on the percentage of the latter metal in the ore.* This loss of lead varies under these and other circumstances from 5 to as much as 15 per cent.

The saving of gold and silver can be put down at 90 to 98 per cent.

The same remarks that apply to concentration, in comparing European and American practice, apply equally to smelting, and the point to determine is how far close work can be carried, to yield most profit on the money invested, in view of the cost of treatment.

The cost of smelting is extremely variable, and depends on the price of labour, fluxes, fuel, composition of the charge, and its behaviour, size of furnace, arrangement of plant, freight and refinery-charges on bullion, and other circumstances.

Under the most favourable conditions as regards ore, fluxes, fuel, etc., the most docile lead ores may be smelted as low as 12s. 6d. to 16s. 8d. per dry ton. Under unfavourable conditions the cost of smelting may run from £1 0s. 10d. to £6 5s., but from £3 2s. 6d. to £5 4s. 2d. is a closer average. In some cases, in dealing with smelting ores it will pay better to concentrate close, and ship and sell the concentrated ore, than to smelt on the spot. In this connexion, it is to be noticed that the degree to which concentration ought to be carried, in preparing an ore for smelting, depends on the value or otherwise of the concentrates and the gangue as a flux. The relative cost of shipping the work-lead to refining works, or of refining locally, has also to be considered.

By far the greater proportion of the ores produced in the silver-lead mining camps of the States are sold under contract or at public auction, on account of the mine owners, to custom smelting establishments, to which sampling works for mechanically sampling the ore are generally attached.

The dry weight, *i.e.*, the percentage of moisture, is usually left to the smelter to determine.

A higher charge is necessarily exacted when the ores are sulphides or highly siliceous. Zinc, sulphate of baryta, and occasionally silica, are

* "Notes on Western Lead Smelting," W. S. Keyes. *Eighth Report of the California State Mining Bureau*, page 804.

charged for at so much per unit, in excess of a certain percentage fixed by the smelter; 5 per cent. more zinc being commonly allowed in oxidized than in sulphuretted ores.

When the ore is calcareous, or highly charged with oxide of iron, a rebate on the ordinary public tariff-price, which represents their maximum charges, is often allowed by smelters. As regards the lead, some smelters require a certain minimum, and an extra charge is made for each per cent. of lead below it; as a rule, no payment is made when the assay shows less than 10 per cent.

It is usual to deduct 5 per cent. for loss of silver in the ore, and calculate the value on New York quotations for the balance; when, however, the ores are dry or siliceous, a still further deduction is made, often as high as 10 per cent. for 100 ounces ore, but less for higher grades.

The gold is paid for from $\frac{1}{10}$th of an ounce per ton upwards, at the rate of £3 15s. to £3 19s. 2d. per ounce. New York quotations for silver and lead on the day of settlement, are assumed as the basis of price.

(*To be continued.*)

THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.*

By A. G. CHARLETON.

Part II.—The Relative Scope of Rival Methods of Ore Treatment.

Having examined the metallurgical nature of the various ores and processes (which have been superficially touched upon in the briefest possible manner), we are in a better position to proceed to the still more complicated and difficult questions of the relative prime cost of plant, working costs, and the percentage of gain or loss in rival methods of ore treatment. These are liable to differ in different countries, and under varying conditions even in the same neighbourhood, according to the nature of the ore, cost and quality of labour, material and supplies, as well as the economical or uneconomical manipulative or mechanical modifications in treatment, which can be introduced in each case, on which probably more than anything else, the individual economy or wastefulness (the profit and loss in fact) in works of the same class will depend.

To form an estimate of the prime cost of plant, we must either know the ordinary gross cost of a mill of a certain size and description in any particular locality, or, what is better, the actual quantities and prices of material of different kinds used, and the number of shifts worked by different classes of artizans at specific wages rates, in the construction of the foundations and superstructure of the building, and the erection of the machinery; as well as the price of the machinery itself, the weight of all the transportable material, and freight rates.

The collection of such facts is unfortunately too rare in mining operations to make it always an easy matter to form estimates of the kind applicable to different localities. It is only by piecing together fragments of information obtainable here and there that one can get anywhere near the truth.

Under the category of rival processes, the relative costs and results of treating the following classes of ore may be compared:—

1. Pyritic gold ores by chlorination in vats and chlorination in barrels, and by the new cyanide process, as well as by grinding (without roasting) in pans and arrastras.

* *Trans. Fed. Inst.*, vol. iv., page 233.

2. Silver ores in general,* by wet and dry crushing and pan-amalgamation, the patio and other processes, as compared with the ordinary and Russel lixiviation processes.
3. Free-gold ores by battery-amalgamation, both with and without grinding, and concentration.

The costs and loss in coarse-concentration and smelting have already been considered, and will not be referred to again.

There is no branch of ore treatment which admits of wider differences of internal detail than wet concentration (producing a corresponding effect on its cost†), but such points cannot possibly be entered into here; and as regards dry concentration, it cannot at present seriously be regarded as a rival to wet methods, and may therefore be passed over entirely.

The concentration of sulphide ores of gold and silver often depends on the relative proportion the sulphides bear to the gangue. No sharp line of distinction can be drawn between ores which are suitable for concentration or not, as average value and local cost of treatment enter into the calculation, but roughly speaking, such ores as contain 40 per cent. of sulphides are not usually suitable for concentration. Frequently an ore which will average 20 to 50 per cent. of sulphides, coming from the mine, is best divided by hand selection into first-class ore (to be treated by smelting, or some other process), and a second and poorer class for concentration.

In a mine where the ore runs in seams and pockets of solid mineral, it may happen that 100 tons of crude ore will contain on an average 20 per cent. of sulphides, which can be separated by hand selection into 20 tons of 75 per cent. sulphides and 80 tons of 6¼ per cent. sulphides. When this can be done cheaply it is often better than subjecting the whole 100 tons to the costs and losses of concentration, whilst it renders commercially possible a more expensive but more efficient process than the original ore could bear.

A point to be remembered, however, in concentration is, that clean, that is to say pure concentrates, are often as important to obtain as

* The term is applied to ores which do not contain more than, say 10 to 15 per cent. of lead. Ores with more than 15 per cent. of lead are known as silver-lead ores, and are smelted when circumstances admit of it, which is the only admissible proposition for their treatment. Carbonate of lead and galena are converted by roasting sometimes into oxy-chloride and sometimes into sulphate; the former goes into the amalgam, whilst the latter does not amalgamate, and this circumstance explains why some bullion of plumbiferous silver ores is free from lead, whilst other bullion sometimes contains 600 parts of lead in 1,000, according to the condition of the lead in the roasted ore.

† In western mining camps, the cost generally runs between 4s. 2d. and 12s. 6d.

clean tailings (which latter are essential for close saving). You may have the one without the other, and although a clean separation of the metallic components of the ore is almost always advisable, in some instances it may be desirable to leave a certain proportion of sand or gangue in the headings, either as a flux for smelting (as previously pointed out) or to lighten the pan-charges in amalgamation.

Vat and Barrel-chlorination.

For working a few tons of concentrates per day the regular Plattner process, with fixed tanks and long exposure to gas, is as convenient and economical as any of the improved processes, because the plant is simple and cheap to erect, and very few hands are required to run it.*

The production of gas is a simple question of relative costs of different chemicals delivered at the mine. For working large quantities of ore daily, especially if of comparatively low grade, which requires economy in crushing, roasting, gas consumption, labour, and time, the use of barrels, in place of fixed tanks, effects a saving in time and labour, and makes special watchful skill less important in the whole process than in the Plattner method, so that one of the modifications of the barrel system would be naturally adopted on crude ore and in dealing with large quantities of concentrates.

The attrition of the ore particles and the thorough turning over of the charge in the presence of nascent chlorine in the barrels, has also some effect upon the results, especially if there be coarse-gold present which has to be amalgamated afterwards, as the gold, it is said, is attacked with much greater avidity by the mercury in consequence of the clean surface created by the action of the chlorine and attrition combined. It is necessary, however, that the tailings should first be washed, by continuing the leaching, till all trace of chlorine has disappeared, and that they should be treated by amalgamation at once. If left to dry, the gold particles assume a red colour and amalgamate with great difficulty.

The most successful furnaces in general use for chlorination-roasting appear to be of three types:—

1. Plain or step-hearth reverberatories furnaces, of the Fortschaufelung class, with a hearth surface of about 12 feet wide by 75 feet long.
2. Horizontal round-hearthed mechanical furnaces, of the English Brunton-calciner type, about 12 feet in diameter.
3. A modified form of the Spence furnace.

* *Gold Amalgamation and Concentration*, page 28.

To which may perhaps be added an altogether new modification of the Fortschaufelung furnace, which will be alluded to later on.

When battery-amalgamation and concentration precedes chlorination, Mr. A. Thies* recommends the use of brass-wire screens in place of slot-punched sheet-iron ones, stating that when using 36-mesh brass-wire in place of 40-mesh slotted-iron, at Haile, in North Carolina, he obtained a far more uniform pulp for concentration, whilst the average life of the wire was found to be six weeks as compared with the sheets, which had to be thrown out in fourteen days, a matter which is worth noting. German practice certainly supports this view so far as concerns the effect of wire screens on the concentration results.

Dr. Egleston states that the correct amount of charge, in roasting gold-sulphides in the reverberatory furnace, is usually 10 to 12 lbs. per square foot of hearth ; an ordinary charge amounting to about a ton at a time on each division of the hearth, of which there are usually three.

A furnace of this description, designed to handle from 3 to 4½ tons of concentrates per 24 hours, 75 feet long by 11 feet wide inside, requires for its construction 40,000 common bricks, 8,000 firebricks, and 5 barrels of fireclay, and the ironwork, tie-bars, etc., exclusive of T-rails, weighs 11,000 lbs. A furnace with a 14 feet by 60 feet hearth requires 86,000 red bricks and 15,000 firebricks, and costs about £626 in Pueblo or Denver.

In vat-chlorination, the rest of the plant to correspond, covered in (as the furnace must also be), consists of a drier, a set of fixed or rotating leaching-vats, with (in some cases) silver-leaching tubs added, precipitating-tanks, and reagent tanks, well and pump, lead-generator, heating-pan and wash-bottle, a drying muffle and melting furnace, and a press and filters for precipitate, as well as filter-tubs for the base metals† in the waste-solutions, and cars for handling the ore, overhead tackle for lifting the vat-covers and tools, etc.

The cost of a plant of the kind in California, with a capacity of 6 tons in 24 hours, is stated‡ to vary between 6,000 and 7,000 dols. (£1,250 to £1,458 6s. 8d.). A small plant would cost about half the former sum.

The Thies modification§ of the Mears process (one of the most successful unpatented systems of barrel-treatment), requires the sub-

* "The Thies Process of Treating Low-grade Auriferous Sulphides."—*Trans. Am. Inst. Min. Eng.*, vol. xix., page 601.

† If present and worth the extra treatment.

‡ "The Milling of Gold Ores in California," by John Hays Hammond. *Eighth Annual Report of the California State Mineralogist*, page 696.

§ A description of this process is given by Dr. Egleston in *The Metallurgy of Gold and Mercury*, page 664, *et seq.*

stitution of rotating barrels for vats, the chlorine being generated inside the barrels, by the action of sulphuric acid (66 degs. B.) on bleaching powder, which is added to the charge of water and ore under ordinary circumstances in the proportion of 10 to 30 lbs. of chloride of lime to about 15 to 30 lbs. of acid per ton of charge of roasted ore in each barrel.

If copper is present more of these reagents are required.

The barrels must be run by steam or water-power, the latter being, of course, the most economical, when available.

The rest of the plant, though differently arranged, is substantially the same as that for vat-chlorination, and will cost not less than £620 for works with a capacity of 5 tons per day; the weight of the machinery will be about 17 tons.

The cost of the vat process at the works of the Plymouth Consolidated Company,* where 100 tons of concentrates is treated per month of thirty days (leaching taking twenty-four days), is given as follows :—

	£	s.	d.	£	s.	d.
Roasting—						
3 men at 10s. 5d. per day for 30 days	46	17	6			
1¾ cords of wood at 17s. 8½d.	46	9	8½			
54 lbs. of salt at 3s. 8d.	2	10	7½			
				95	17	10
Chlorine (employing two generators)—						
60 lbs. of manganese per day at £9 15s. 10d. per ton	7	1	0			
68 lbs. of salt „ 3 2s. 6d. „	2	11	0			
120 lbs of acid „ 12 10s. 0d. „	18	0	0			
				27	12	0
Leaching—						
40 lbs. of sulphuric acid for settling-tanks for 24 days	6	0	0			
40 lbs. of sulphuric acid for making sulphate of iron for 24 days	6	0	0			
Wages of leachers (2 men at £1 2s. 11d.†) for 30 days	34	7	6			
Wages of foreman	26	0	10			
				72	8	4
Total cost per month ...				£195	18	2

or £1 19s. 2d. per ton.‡

* *Trans. Am. Inst. Min. Eng.*, vol. xv., page 305.

† Two men on day shift attend to all the work of handling the ore after it is leached. The head man receives 12s. 6d., the others 10s. 5d. Only three tankfuls are leached every four days; but the men are employed steadily. The sulphate of iron is manufactured on the spot.

‡ To this should be added for assays, repairs, supplies, insurance, taxes, water, interest on invested capital, and deterioration of plant, as well as proportion of general expenses and superintendence, about 16s. 8d. additional per ton, making the total cost of working £2 15s. 10d. per ton.—*Eighth Annual Report of the California State Mineralogist*, page 48.

At the Providence Works* two furnaces are used, which have a capacity of 9 tons in 24 hours. Each furnace consumes 1 cord of wood, and the cost per day can be reckoned as shown in detail on page 58. This makes the cost of treatment per ton of sulphides to be 14s. 9½d when the works are run at full capacity.

As the ore, however, only contains about 7 per cent. of sulphides, or 4⅓ tons in the 62 tons milled daily, this quantity does not keep the two furnaces running full time, though both of them are in constant operation. As most of the expenses remain the same, whether running at full capacity or not, the actual cost of chlorination, figured on a working basis of 4½ tons treated daily, approximates, therefore, more nearly to £5 13s. 10d. per day, or £1 5s. 3½d. per ton of concentrates, as detailed below. Add to the above sum (£5 13s. 10d.) the cost of milling per day, viz., £11 19s. 7d., and it will be found to represent a total outlay of £17 13s. 5d. per day, or a total charge of 5s. 8½d.† per ton of crude ore raised from the mine.

This estimate makes no allowance for general supervision, interest on first cost, or deterioration. The conditions of treatment in these works are, however, very special, and can hardly be considered as a basis to estimate upon in most instances.

The estimated cost of the treatment of 4½ tons per day is:—

	£	s.	d.
Labour	2	13	1½
2 cords of wood at £1 0s. 10d.	2	1	8
14 lbs. of manganese at 1½d.	0	1	7
126 lbs. of salt at ¼d.	0	5	3
104 lbs. of acid at 1d.	0	8	8
Lime, sulphur, and calcium hyposulphite ...	0	0	7½
Illuminating	0	0	10
Extras	0	2	1
Total cost per day ...	£5	13	10

or £1 5s. 3½d. per ton.

At the Alaska Treadwell Company's works the cost for the year ending May 31st, 1891, was as follows, calculated on a basis of 3,568 tons of concentrates treated:—

* *School of Mines Quarterly*, vol. v., and Eglcston's *Gold*, page 657.

† 1s. 10½d. of this is chargeable to the chlorination.

	Cost per Ton. (American Money.) Dollars.	Cost per Ton. (English Money.) £ s. d.	Gross Cost. Dollars.	Gross Cost. £ s. d.
Labour—				
Foremen...	·5600	...	3,006·43	...
Engineers and foremen	·0534	...	286·38	...
Gas generator	·2089	...	1,121·29	...
Salters	·0855	...	458·24	...
Floormen	·7604	...	4,080·83	...
Roasters	2·9400	...	15,782·49	...
Carmen	·2077	...	1,114·72	...
Carpenters	·0039	...	21·39	...
Labourers (white) ...	·0999	...	536·62	...
„ (Indian) ...	·0309	...	166·09	...
Teamster	·0806	...	433·11	...
	$5·0312	£1 0 11¼	$27,007·50	£5,626 11 3
Supplies—				
Acid	1·0220	...	5,486·38	...
Wood	1·5230	...	8,175·93	...
Coal	·0543	...	284·80	...
Furnace supplies ...	·0270	...	145·19	...
Manganese and salt ...	·8095	...	4,345·86	...
Pipe	·0089	...	47·90	...
Generator fittings ...	·0038	...	25·93	...
Leaching tanks... ...	·0638	...	339·05	...
Miscellaneous	·1353	...	727·26	...
Rake-heads	·0360	...	193·50	...
Car wheels	·0037	...	27·50	...
Repair account... ...	·1352	...	720·08	...
Electric light account...	·1077	...	578·49	...
Haulage account ...	·0549	...	294·77	...
	$3·9852	£0 16 7¼	$21,392·63	£4,456 15 11¼
Total cost per ton of sulphides ...	$9·0164*	£1 17 6¼	$48,400·13	£10,083 7 2½
			Total gross cost.	

* In the six months ending November 31st, 1892, this charge was reduced to $8·42 = £1 15s. 1d. per ton of sulphides. As an instance of economical management, the total costs of mining and milling per crude ton extracted may be cited as follows, the ore being quarried in open benches:—

	Cents.		s.	d.
Mining	65	=	2	8½
Milling and concentrating	33	=	1	4½
Chlorination	19	=	0	9½
General expenses (mine)	8	=	0	4
General expenses (in San Francisco)...	2	=	0	1
Bullion freight, etc.	5	=	0	2½
Total cost ...	$1·32	=	5	6 per crude ton

of ore, but the average for some time previous was 6s. 3d. per ton.

The roasting at these works is done in Spence automatic furnaces, each with four hearths, which roast 8 tons a day. On the hearth next the lower one, 3 per cent. of salt is added with a special spoon. The ore, instead of being as formerly stirred by fixed rakes, is now rabbled by oscillating ones, which tip every three minutes. On the upper drying-hearth they last a long time, but on the lower ones, where the heat must be great to drive off the last traces of sulphur, their life is only about three months.

Formerly the rakes stopped between the forward and backward motion just over the flue, now they are made to go beyond it, and are no longer exposed, as they were at this point, to the sulphurous gases and hot falling ore, which clogged the end of the shelves and gave such trouble at the Haile mine, finally causing the abandonment of the Spence furnace there for roasting fine ores, although admittedly a most excellent mechanical roaster for coarser grades of material in its original shape.

With the important modification above alluded to it seems doubtful (judged by the above facts) how far Mr. Thies' condemnation of the Spence furnace* (in works with a capacity of 3 to 4 tons) for fine ores (where a dead sweet roast is required) is entirely justified.

The labour involved in vat-chlorination on an average may be taken at :—

1 man, who works 10 hours, to bring in wood and ore.
1 man at the furnace, who works 8 hours.
1 chlorinator, „ 8 „
3 labourers (1 on shift by day, and 2 by night).

At Sutter Creek it takes 8 men for the whole 24 hours, one chlorinator, one helper, five men at the furnace, and one who wheels ore.

The consumption of chemicals per ton of ore is about as follows :—

Salt in the furnace	40 lbs.
„ „ generator	6 „
Manganese „	6 „
Sulphuric acid generator (66 degs. B.) ...	20 „

The stock of chemicals ordinarily kept on hand, on the scale of operations of the Alaska Treadwell Company, seems to be :—

	£ s. d.		£ s. d.
Salt, about 53 tons, valued at	2 17 11	...	153 9 7
Manganese, 13 „	9 15 10	...	127 5 10
Sulphuric acid, 35 tanks, at	8 18 10½	...	313 0 7½
Total value			£593 16 0½

* *Trans. Am. Inst. Min. Eng.*, vol. xix., page 610.

The works in Grass Valley, where vat-chlorination is most extensively used, guarantee an extraction of 90 per cent. of the gold, and 60 per cent. of the silver, but the saving more often amounts to between 90 and 94 per cent. of the gold in pyritic-concentrates, and over 60 per cent. of the silver, if the tailings are leached to obtain it.

The ore to be properly roasted ready for leaching should maintain a nearly vertical face when made into heaps on the finishing hearth, and cut down with a spadelle. It should show no bright specks, but be inclined to become black, which will generally be in 7 or 8 hours. The largest amount of gold has been shown to be in the best condition to be leached and to consume the least amount of chlorine in chlorinating it, when the ore falling in the furnace, in turning it over, has a slight violet colour. When the ore sparkles, and the sparks are numerous and bright, it shows that the roasting is not properly finished, and more salt has to be used.*

A ton of roasted ore will occupy usually 24½ cubic feet. This quantity is derived from 2,800 lbs. of raw sulphides, which occupy about 13⅔ cubic feet per ton (2,000 lbs.). A ton of sulphides will weigh from 1,450 to 1,700 lbs. after roasting, and occupy about 17½ cubic feet. The roasted ore ought not to contain more than 1⅛ per cent. of sulphur.†

At the Plymouth Consolidated works, the Fortschaufelungs-ofen used for roasting is 12 feet wide by 80 feet long, including the fire-box, the hearth being a continuous plane, but the charges, of which there are three in the furnace at one time, are kept entirely separate. They are called by the furnacemen the drying, burning, and cooking-compartments. In the middle division the ore is spread out very thin, and occupies about double the space of either of the others.

The furnace is worked by 8 hours' shifts, a charge being drawn and added in each shift. The charges weigh 2,400 lbs. and carry about 10 per cent. moisture.

The ore averages about 20 per cent. of sulphur, and just before the sulphur ceases flaming in the second division of the furnace 18 lbs. or ¾ per cent. of salt is added to the charge. Care must be taken to keep the concentrates damp until they are introduced into the furnace, or a decomposition of the pyrites begins, forming lumps which do not roast, and

* The heat on the finishing hearth must be maintained at a lively bright red, but not at a white heat, else the gold particles would melt, which, with a good magnifying glass, can be easily detected; after washing off the iron the gold appears then in minute globules, the chlorination of which is more difficult.

† Egleston, *Gold*, page 622.

consequently cause a loss of gold in the residues from leaching. The roasted ore from each shift is kept by itself on the cooling-floor until a tankful (about 4 tons) has accumulated from a single man's shift, and it is then worked by itself. This enables the foreman to better control the work of roasting, for if one lot out of three works badly, it points to the fault being with the furnaceman; whereas, if all three give unsatisfactory results, it may be presumed to be owing to a change in the ore, and the roasting must be modified.

The vats* for chloridizing the roasted ore are 9 feet in diameter by 3 feet in height; they are four in number, and are slightly inclined forwards to drain them completely. The bottom of each tank is composed of a filter about 6 inches thick, consisting of light strips of $\frac{3}{4}$ inch wood laid on the bottom at intervals of about a foot. Across these are placed 6 inch boards, spaced an inch apart. On this loose floor coarse lumps of quartz are spread, and on the top of this again quartz-sand, until a depth of 6 inches is obtained. Finally, this sand filter is covered over with another loose floor, the boards lying cross-wise to the loose floor beneath and pretty close together. This upper floor is merely to facilitate shovelling the charge out when it has been gassed to permit the removal of the leached ore without disturbing the filter.

The ore to be chloridized must be damp (about 6 per cent. moisture). The working test is to take a handful of ore and squeeze it, then open the hand, and if the lump immediately begins to crumble and fall apart (not run) the ore has the requisite amount of moisture. The damp ore is screened into the tank so as to lie as loose as possible and thus facilitate the penetration of the chlorine. A coarse screen of one-and-a-half mesh is used for this purpose.

The tanks are only filled up to within about 3 inches of the top, (to ensure the entire contents being covered with water in the subsequent leaching,) otherwise there would be great difficulty in washing out the soluble gold. As soon as they are full they are gassed. The gas is introduced into the bottom from two opposite sides, and is continued until ammonia held over the ore gives off dense fumes of ammonium chloride, which usually happens in about 4 hours. When this point is reached, covers are placed on the tanks and luted on.†

* The vats should be coated on the inside with asphaltum varnish or a mixture of pitch and tar applied hot, but the former is preferable, as it penetrates better into the pores of the wood.

† It is usual to provide the covers of the tanks with two pieces of $1\frac{1}{4}$ inches gas-pipe 6 inches long, and a square hole, 6 inches by 6 inches, closed with a wooden cover. These pipes are closed with balls of clay during the impregnation of the ore

The two gas-generators* which are employed to charge a tank are allowed to work on till nearly exhausted, when they are disconnected, and the holes in the tank are plugged up. The tanks are usually charged in the morning, and left standing two days. On the third day the ore is leached. The tank is first filled with water, and allowed to stand a few minutes to permit the water to penetrate the ore. If no more water is absorbed, the liquor is drawn off at the bottom, care being taken to keep the tank full of water during this part of the operation, which lasts 4 to 5 hours.

For charging the tank a gunny sack is laid on the ore and held down with a couple of bricks where the wash-water is afterwards to be introduced, in order to better distribute the water in the tank, and prevent it washing out holes and packing the ore.

The liquor from the leaching-vats is conducted to settling or storage-tanks, where about 40 lbs. of sulphuric acid is added (66 degrees B.) and it is allowed to stand 2 to 24 hours. It is then run into the precipitating-vats, where the gold is precipitated with sulphate of iron; the iron solution being added until after stirring, a further addition produces no purple colour. After the gold is precipitated it is allowed to stand from two to three days to settle, when the supernatant liquor is drawn off with syphons into a second settling-tank, where any gold drawn off by the syphons has a second opportunity of settling.

The liquor stands in this tank till it is necessary to run it off to make room for a fresh charge. Very little gold is found in this tank, so it is only cleaned out about once a year.

In the meantime fresh liquor has been run into the precipitating-tanks on the gold already precipitated there. In this way the gold is accumulated till the semi-monthly clean-up, the precipitating-tanks being kept locked and covered. In making a clean-up, the supernatant liquor is syphoned off, the gold gathered up, and placed in a filter of punched iron, lined with filter-paper, and washed with water till all the acid and iron salts are removed. It is then dried, melted in crucibles, and cast into bars.

Under unfavourable conditions the cost of the chlorination process may run up as high as £4 3s. 4d. per ton, but in California it may

with gas. After gassing, the clay is removed, and one of the pipes is coupled to the hose of the water-tank, whilst the other is connected either with another tank ready for chloridizing, or the ashpit of the roasting furnace, partly to utilize the surplus chlorine, as well as to protect the workmen from its injurious effects.

* When practicable, it is a good plan to heat the generators by steam in place of direct heat.

generally be taken as costing £1 17s. 6d. to £3 1s. 6d. per ton, £2 14s. 2d. being in all probability a fair average on a basis of treating 3 tons or more daily.

Mr. G. F. Deetken is responsible for the statement* that with favourable conditions of cheap wood and freight rates, not counting interest, insurance and amortization of a capital of 3,000 dollars, the working expenses in some parts of California do not exceed £1 0s. 10d., and are sometimes as low as 16s. 8d. per ton.

In locating works of this kind it is important to secure a good fall, and they should be terraced and so placed with reference to the prevailing winds that noxious fumes will not be carried in the direction of valuable land or house property.

There should also be a supply of water at hand, delivering 40 to 60 gallons per hour.

Barrel-chlorination.

Mr. Thies, in a letter to Mr. C. N. Aaron (a well-known Californian authority), gives the cost of the Thies process at the Haile mine, North California, as follows :—

Using a double reverberatory furnace, which furnishes 2 tons of roasted ore every 24 hours, with an average consumption of 1 cord of wood, at 5s. 2½d. per cord, and employing four labourers, the cost of roasting the ore amounts to 10s. 11¼d. per ton. Two men can easily treat 4 tons in 10 hours, elevate the ore, and clean out the filter-tanks, of which there are four to each barrel. Arranged on this basis, the cost of roasting and chlorinating amounts to 19s. 3¼d. per ton, as tabulated below :—

	Cost per Ton. s.	d.	s.	d.
Roasting 2 tons of ore—				
4 labourers at 4s. 2d.	8	4		
1 cord of wood at 5s. 2½d.	2	7¼		
			10	11¼
Chlorinating 4 tons of ore—				
2 labourers at 3s. 9d.	1	10½		
1 chlorinator at 8s. 4d....	2	1		
40 lbs. of bleaching powder at 1½d. ...	1	3		
60 lbs. sulphuric acid at 1d.	1	3		
Sulphuric acid, for making ferrous sulphate	0	6¼		
Repairs, wear and tear...	0	10		
Power	0	6¼		
			8	4
Total cost per ton			19	3¼

* *Eng. and Min. Jour.*, New York, vol. lv., page 244.

We thus have the sum of 19s. 3½d. for roasting and chlorinating 1 ton of roasted ore, representing 1⅓ tons of raw iron pyrites.

Inside of 7 hours from the time the ore is in the chlorinator, the solutions are ready for precipitation and the tailings are clean.

At the Phœnix mine, in North Carolina, the cost for roasting* and chlorinating by the Thies process is estimated at :—†

	£	s.	d.
Roasting	0	9	1
Chlorinating	0	11	1
	£1	0	2

2,353 tons of concentrates are said to have been successfully treated by this process at the Haile mine and 5,000 tons at the Phœnix mine between May, 1888, and September, 1890.

At the Phœnix mine, a 12 feet revolving pan-furnace is used, which roasts 1 ton of raw ore in 12 hours, with a consumption of ⅜ths of a cord of wood, and 3s. 9d. worth of labour. The cost of power does not exceed 1s. 0½d. per ton.

At the Bunker Hill works, the concentrates are roasted with 1 per cent. of salt, in a reverberatory furnace 40 feet by 12 feet on the outside, with walls 18 inches thick, the stationary hearth of which (7 feet by 18 feet by 2 feet in height, with a working door on each side) terminates in a horizontal revolving hearth, 12 feet in diameter, set at a lower level (giving a fall of 6 or 7 feet). The discharge hole or cub is in the centre of this latter hearth, from which the ore drops into cars.

At the Deloro mine,‡ Canada, where the Mears process was formerly in operation, the ore was roasted (after drying in a revolving drier) in a pair of cylinders,§ one 30 feet in length by 5 feet in diameter, the other 20 feet by 4 feet, set tandem, jacketed with mineral wool and then paper. It is claimed that 10 tons of concentrates have been roasted in these cylinders in 24 hours, so that the extraction of gold has reached 93 to 98 per cent.

The tanks used in the process should be built of wood which has been soaked in linseed oil, dried, and painted with three good coats of white lead or tar.

* Removing the sulphur to within 0·25 per cent.

† Phillips, *Trans. Am. Inst. Min. Eng.*, vol. xvii., page 321.

‡ R. P. Rothwell, *Trans. Am. Inst. Min. Eng.*, vol. xi., page 191.

§ Different furnaces require, on the average, it is said, the following weights of wood to roast a ton of ore (the weights being calculated on the basis of 2,200 lbs. per cord as the average) :—White furnace, 300 lbs.; Bruckner, 900 lbs.; reverberatory, 600 lbs.; Stetefeldt, 200 lbs.—(Report, *Tenth Census, United States.*)

Mr. Thies states that the cost of barrel-treatment depends chiefly on the number of tons chlorinated per day.

The wear on the inner lead-linings of the chlorinators is imperceptible; a chlorinator in use at the Phœnix mine for over five years does not show any wear. The lining is fastened on with bolts from the outside, as it has been found very difficult to burn on lead, when chlorine has acted on it, in making repairs.

At the Haile mine, the fire-assay and value of the ore delivered to the stamps is 4·50 dollars (18s. 9d.) per ton.

The mint returns of bullion gave 16s. 3d. per ton of ore treated, of which 6s. 0½d. is to be credited to the battery, *i.e.*, to free-gold; whilst 10s. 2½d. was obtained from the sulphides.

Taking the assay-value of the ore at 18s 9d., and the actual yield in bullion at 16s. 3d., there is an indicated loss of 2s. 6d. per ton, or 13⅓ per cent. Taking the yield in free-gold at 6s. 0½d. per ton from an ore worth 18s. 9d. per ton (or approximately 32 per cent. of the gross value by assay), we have 68 per cent. to be debited to the concentrates. But the total yield in free-gold, and in gold from the sulphides being 16s. 3d., the ratio of the free-gold saved, to the total amount saved, is approximately 38 per cent., and the combined gold 62 per cent., or a saving of about one-third free and two-thirds combined gold.

The term, combined gold, has been used to express the condition of the gold which is not free. Whether the gold that is not free is chemically or mechanically diffused in the sulphides or both is uncertain, in most cases.

Mr. T. W. T. Atherton* claims to have found gold as a natural sulphide in the pyrites of the Deep Creek mines of New South Wales. He gives an analysis of the ore and the method pursued in experimenting on it, from which his conclusions were drawn.

From 80 tons of ore stamped per day of 24 hours at Haile, 7½ tons of concentrates are obtained from 16 Embrey end-shake tables, which gives as the yield of each concentrator a little less than half a ton.

The average assay-value of the raw concentrates for the 12 months preceding the date of Mr. Thies' paper was £6 5s. per ton, and the percentage of sulphur they contained varied between 40 and 45 per cent. In roasting, this was brought down to 0·25 to 0·40 per cent., whilst the value of the material naturally increased; the raw concentrates worth £6 5s. becoming worth when roasted £8 6s. 8d., or about one-third more per ton.

* *Eng. and Min. Jour.*, New York, vol. lii., page 698.

The assay-value of the tailings thrown away, after chlorinating the roasted ore, was, on the average, only 8s. 4d. per ton, representing an extraction of 95 per cent.

The process had been in successful operation 2½ years, at Haile, when Mr. Thies' paper was written in 1890, and 36,000 tons of crude ore had been treated profitably during that time, prior to which nearly every process had been tried on the ore and failed.

The advantages of the process are:—The small amount of space the plant occupies, speed of operation, high percentage of yield, facility of ascertaining the condition of the charge at any time,* and very slight wear and tear. The only offsets against these advantages are the care and intelligence required to control it, and the need for a small amount of power.

250 to 300 gallons of wash-water are needed per ton of ore treated (an ordinary charge), the water being introduced into the barrel first.

It takes 2 to 4 hours to chlorinate each charge, the barrel revolving at 12 revolutions per minute.

The lead-lining is ¼ inch thick, and weighs 12 lbs. per square foot.

The filters must be flooded from below with 4 or 5 inches of water to prevent the ore packing as it falls into them.

The charge must be allowed to drain, and then washed with water as rapidly as possible till the wash-water shows no gold, though still carrying traces of chlorine.

If lime is present, settling-tanks, called stock tanks, are required to settle the liquors which are drawn out of them into the precipitating tanks after standing for 14 to 16 hours.

The precipitating-tanks are large enough to hold the liquors from 8 tons of washed ore. The ferrous sulphate is syphoned into them. The solution must have a decidedly acid re-action in order to be certain that all the lime has been converted into sulphate.

The Providence works and those of Bunker's Hill may be cited, the former as showing exceptionally low cost by the vat process, working 9 tons per diem; the latter unusually high cost by barrel-treatment, treating 2 tons in 24 hours. To compensate for this the loss in the tailings, which used to run as high as £1 9s. 2d. when worked by the ordinary vat process, has been reduced 50 per cent.

* For this purpose a lead valve is arranged in the barrel, so that not only the pressure of an excess of chlorine gas but its actual presence can be ascertained at any time. It does not do to trust to the pressure test alone, other gases being some times given off.

The cost at the Providence works :—

	£	s.	d.
1 foreman	0	12	6
1 white labourer	0	9	4½
5 Chinamen at 6s. 3d.	1	11	3
2 cords of wood at £1 0s. 10d	2	1	8
29 lbs. of dioxide of manganese at 1¾d.	0	3	4
260 lbs. of salt at ¼d.	0	10	10
216 lbs. of sulphuric acid at 1d.	0	18	0
Lime, sulphur, and calcium hyposulphite	0	1	3
Illuminating	0	0	10
Extras	0	4	2
	£6	13	2½
Per ton ... ... 14s. 9½d.			

The cost at the Bunker Hill works :—

	s.	d.	£	s.	d.
Roasting—					
2 roasters at 13s. 6½d.	13	6½			
⅛ths cord of wood at £1 5s.	15	7½			
			1	9	2
Chlorinating—					
1 chlorinator at 12s. 6d.	6	3			
30 lbs. bleaching paper at 2d.	5	0			
36 lbs. of salt at ¼d.	0	7½			
20 lbs. sulphuric acid, 66 B. at 1¼d.	5	3			
Water-power	2	1			
General expenses and loss	12	6			
			1	11	8½
Total cost of 1 ton of ore roasted and chlorinated ...			£3	0	10½

Under efficient management and favourable conditions the cost of barrel chlorination will usually be found to vary between 12s. 6d. and £1 0s. 10d. per ton in America.

The results of treatment by another vat-process, the Pollock patent, are given in the *London Mining Journal*,* treating various ores as follows :—

	Ozs.	Dwts.	Grs.	Percentage Extracted.
Sheba mine tailings...	2	7	21	96
Mixed lot of ore, Transvaal	1	3	22	97
Day Dawn P.C. quartz, Q. 6	6	9	23	97
„ concentrates, Q. 1	1	2	6	97
Swaziland quartz, South Africa	0	16	5	95
City and Suburban quartz, Transvaal	12	10	10	95
Crown ore, New Zealand...	3	16	0	94
Transvaal Gold Extracting Co.	1	10	11	93
„ „ tailings	0	7	8	85
Mount Shamrock tailings	0	8	17	96

* *The Queensland Mining Journal*, Nov. 5th, 1888.

It is stated that the cost of the plant for treating 100 tons a day is about £10,000.

It is certain that ores like those of Meadow Lake district, California, composed of a species of siliceous hæmatite (which in depth will probably be found in an unoxidized condition, consisting mainly of pyrite) will not amalgamate properly, and attempts have been made with various devices and countless processes to work them, but so far without success.

They are said to average £1 9s. 2d. to £2 1s. 8d. per ton or over, and to bear a resemblance to the ores of Bald Mountain district, South Dakota. It is possible that if a sufficient quantity of concentrates could be secured to keep works of the kind running many of these ores could be advantageously handled by the barrel-chlorination process, without a previous roasting, if treated with an oxidizing agent such as nitre cake, and in working the surface-ores even concentration might at first be dispensed with.

An interesting modification of the barrel process (described by John E. Rothwell, *Eleventh Census*, United States, 1890) is to make the chlorination-barrel also the washing and leaching-vessel. This is effected by fixing a diaphragm as a filtering medium, so as to form the chord of an arc of the interior of the barrel. The diaphragm or filter is made up of plates corrugated similarly to the ordinary filter-press plate, and perforated with holes every 4 or 6 inches square. These plates are supported on bearings bolted to the shell. On the top of the corrugated plates is placed the filtering medium—an open-woven asbestos cloth. It is about as coarse as the common gunny-sack, but the warp and woof are of much heavier thread.

Over this is placed an open grating, and the whole is held in place by cross-pieces, the ends of which rest under straps bolted to the inside shell. In this way, though the whole is rigidly held in place, it is very easily and quickly removed when necessary to change the asbestos cloth. Two valves on each end of the barrel above and below the filter, are provided for the inlet and outlet of the wash-water and solution respectively.

The barrel is charged by first filling the space under the filter with water, which at the same time is allowed to pass through the filtering medium and wash it, then the required quantity of water is put in above the filter. There are two methods of charging the pulp, lime-chloride, and sulphuric acid. In one the lime is so placed in the ore-charge in the hopper over the barrel that it goes in with the ore and is completely buried with it. The acid can then be added with very little danger of generating any gas before the plate on the charging-hole can

be put on and securely fastened. The other way, which seems still better, is to pour the acid first into the water, through which it sinks to the bottom in a mass, and does not mix. The ore is then let in, and the lime added last. The chances of generating any gas are stated in this way to be much less. A barrel so charged has been known to remain open 5 to 10 minutes after charging without generating gas, but it has been demonstrated that on the first revolution of the barrel the gas is at once liberated and creates considerable pressure. After the chlorination is complete the barrel is stopped, so that the filter assumes an horizontal position. The hose is attached to one of the outlet pipes, and conducts the solution to the reservoir tank. A hose is also attached to the inlet pipe, water is pumped in under pressure, and the leaching commences.

The air in the top part of the barrel is compressed, and forms an elastic cushion, which gives the wash-water perfect freedom to circulate over the whole surface of the charge, and wash every portion thoroughly with the smallest quantity of water possible. By washing in this way no gas is allowed to escape into the building. The solution runs into a covered reservoir-tank from which an exhaust fan draws the excess of gas, and discharges it outside the building. The length of time needed to do the leaching, varies with the leaching-quality of the ore. Charges have been leached in 40 minutes with a pressure of 30 to 40 lbs. per square inch. With higher pressures the time can be materially shortened. As can be readily seen, the ore in the barrel is in the best possible shape for rapid and perfect leaching. When the barrel is stopped, the ore settles on the filter, the coarsest and heaviest on the bottom, graduated evenly over the whole surface up through the charge to the slimes on top. In order to facilitate the leaching of charges carrying an excess of dust, a valve placed in the casing of the head (on a level with the surface of the pulp) is opened just after the barrel is stopped, and the slime remaining in suspension is run off into an outside washing filter-press, where it can be treated separately, and the charge washed in the usual way. The tailings are discharged into a car which will hold the whole charge of ore and water and then run out, or if water is abundant they are discharged into a sluice and washed away.

For leaching purposes the amount of water needed to wash a charge varies very little with the richness of the ore, going to show the perfect leaching-condition of the ore in the barrel. The amount required is about 120 gallons per ton more than the quantity used in the barrel for chlorination, which is about 100 gallons per ton.

In order to get a concentrated solution for after-treatment, and to

reduce the amount of solution to be treated, a tank is placed over the barrel, and when the richest of the solution and wash-water has run out into the reservoir-tank the discharge hose is connected with a pipe leading to the upper tank, and the washing is finished into it. The solution collected in this way is used in the next following charge in the barrel. The quantity of solution to be precipitated is thus reduced to 120 gallons per ton of ore treated.

The advantages claimed for this method are: (1) the freedom of the building from chlorine gas; (2) the control obtained over the perfect washing of the ore; (3) the small amount of labour, especially skilled labour, necessary; and (4) the small amount and simplicity of the machinery for the great amount of work accomplished.

One man of ordinary intelligence and a helper, are able to look after three barrels—charging, leaching, and discharging them. If the tailings are sluiced out no extra help is needed, but where they have to be trammed, one man in addition is necessary. The disadvantages are due to the necessary construction of the barrel, but do not interfere with its successful working. They are principally the amount of space taken up by the filter and the portion of the barrel underneath, and the fact that when the barrel is charged and running it is not perfectly balanced.

These disadvantages can be partly overcome by placing the filter close to the shell, only leaving sufficient space underneath to allow of free circulation, but bringing it up to the same height on the sides of the barrel as the horizontal filter; then, by using compressed air to displace the solution and wash-water, an equally good result can be obtained.

For the collection of the solution two tanks are necessary, each of ample capacity to hold a day's solution from all the barrels. Those for collection are placed on the same floor as the chlorinators (unless fall can be secured to place them below). On a step below are the precipitation-tanks, which should be of the same capacity and number as the collecting-tanks. The limit to size would probably be 50 tons capacity; where more is treated, another battery of tanks would be needed.

For a precipitant Mr. Rothwell recommends hydrogen sulphide gas generated from paraffin and sulphur or from iron sulphide and sulphuric acid as the cheapest and most satisfactory. It is generated, and then forced through the solution with a small air-pump, which at the same time forces air through, keeping the solution-tank in an agitated state and expelling any free chlorine gas. To save time the gas is turned into the tank while it is filling up, so that, when the tank is full, a few minutes finish the precipitation and collection. The tank is now allowed to stand two or

three hours, when it will have settled sufficiently to allow of the supernatant liquor being drawn off through a filter-press. There is little danger of precipitating arsenic and antimony which may be present when the process is worked cold, as they do not commence to come down till some time after the gold has been precipitated and collected.

This precipitant would not be, however, desirable with any considerable quantity of copper or lead in the solution, but small quantities can be dealt with in the after-treatment.

The loss in gold is considerably less if the precipitate is allowed to accumulate in the tanks, and a clean-up made after six or ten precipitations, than if it were filtered through a press and collected after every precipitation. There does not seem to be any advantage derived from a continuous precipitation and collection on this account if hydrogen sulphide gas is used, as the filters will soon become so coated and clogged with the sulphides as to retard rapid filtration without extreme pressure, which is sure to increase the loss.

The handling of a large number of filter-cloths is also a source of loss, no matter how carefully done. The asbestos filter-cloth under ordinary conditions will last 100 charges, and can be changed in about an hour and a half; one cloth has been known to last 150 charges.

The life of the supporting plates and grating can be made to equal the life of the lining of the barrel, and with barrels that have several thousand charges to their credit this shows little wear.

The latest suggested modification of the chlorination process appears to be that patented by W. D. Bohm, described and illustrated in the *New York Mining Journal* of December 31st, 1892. Briefly stated, the principles involved in it, are a forced upward circulation of the solvent solutions through the powdered ore placed in a suitable vat, of special construction.

The circulation is maintained until the precious metals are dissolved, when air-pressure is applied above the charge, to force out as much of the solution as possible, wash-water being subsequently forced up from below, or admitted above the charge, and then forced out in a similar manner, the previous constant flow upwards having caused (it is stated) such a deposition of the sand as to allow the liquid to be expressed rapidly and cleanly. It is claimed by using a solution of chlorine in water, and circulating it in closed pipes and vessels, that a considerable saving of gas is effected, and that as rapid results are obtained as with agitation, with much less power and wear and tear, while ores of different character can be treated in a short time and with little labour.

The plant is said to be equally applicable to cyanide treatment; the

precipitation of the gold being effected on shavings of an alloy of zinc and sodium is claimed to be more efficient than zinc alone. The chlorine water is circulated through the ore in a manner similar to a cyanide solution. It is claimed for the electrolytic chlorine-producing apparatus, the invention of two Russian chemists (which is worked in connexion with this process), that a great reduction in the cost of producing chlorine is effected; that common salt will supply the place of sulphuric acid and chloride of lime, that the machine is simple, requiring no skilled labour, and has been running with success at the El Dorado mine, Siberia.

This plant, in charge of an engine-driver, produces 40 lbs. of chlorine gas from less than 150 lbs. of common salt, utilizing 5 horse-power. The results are vouched for by M. Leon Perret, mining engineer to the Imperial Russian Government.

A diaphragm-pump of special construction prevents any emulsified grease or other undesirable matter from getting into the solution, and stops leakage of the gold solution through defective glands, etc.

The cost of working this process is stated to vary with the locality. Kaolin ores of Mount Morgan have been treated before the erection of the electrolytic chlorine generator at the following cost:—

	s.	d.
10 lbs. of bleach, at 4 cents	1	8
8 „ acid, at 2 cents	0	8
Labour and power	1	5½
	3	9½

The time occupied for each charge being 4 hours.

The extraction is not stated, but a proportionate increase in cost of chemicals would be necessitated in the treatment of ores requiring more chlorine.

Mr. Claude Vautin mentions, as recent useful innovations, glass-lined iron pipes, manufactured by Messrs. D. Rylands & Co.'s enamelled iron diaphragm-pumps for conveying corrosive solutions, made by Messrs. Scott & Sons; and a compound containing upwards of 75 per cent. of available sulphuric acid, solid below the temperature of boiling water, which can be packed in iron drums, and shipped as ordinary cargo (obtainable from the Ore Dressing Co., 42, Old Broad Street); also chloride of lime packed in specially prepared iron drums. The form in which these two latter reagents are sold may be of utility to persons employing the chlorination process, especially in localities distant from a railway where transport has to be taken into account.

Mr. T. R. Rose,* commenting on Mr. Vautin's statement "that precipitated sulphide of copper is not to be recommended as a practical precipitant for gold, owing to its physical condition, and the facility with which it is oxidized to sulphate of copper," says:—"In accepting this statement as correct in a certain limited sense, it must not be forgotten that the suitability of the precipitant depends mainly on the manner in which it has been prepared. By adding a boiling saturated solution of sulphide (or rather a mixture of the polysulphides of sodium) to the equivalent proportion of a boiling saturated solution of sulphate of copper with vigorous stirring, sulphide of copper is precipitated in a granular form, which settles quickly in water. This substance is more active in the precipitation of gold than the fused sulphide. Its activity is due to the larger surface presented by the porous granules, and to the fact that it is easily decomposed. It is, as Mr. Vautin says, somewhat liable to atmospheric oxidation, but the loss, owing to this cause, is insignificant, because it is always kept covered with water, and can only take up such oxygen as may be dissolved in the latter. The granules of the sulphide of copper can be made the size of peas, under favourable conditions, and there is no practical difficulty met with in precipitating gold rapidly and completely by their use. A point in favour of the fused sulphide of copper is the readiness with which it can be obtained, and as it has been proved to answer well, there is no reason why it should be rejected in favour of the precipitated salt."

The Cyanide Process.

The plant required for the MacArthur-Forrest process consists of crushers (stamps or rolls) to pulverize the ore to 20 to 60 mesh, and leaching and precipitating-tanks of much the same description as those used for chlorination; but as its professed object is the treatment of the ore in bulk, the number of tanks, etc., required for this method of treatment are of necessity much larger than in chlorination works. The cost of erection of such a plant on a basis of treating 50 tons per day would not, so it is said, ordinarily exceed £6,250, everything included.

The report of Mr. J. R. Bradshaw (assayer, etc., of Charters Towers) states, without going into details, that from information he received at Ravenswood, North Queensland, it can be worked there to profit at a cost of £2 per ton, but the cost must of necessity depend on the district and surrounding circumstances.

Mr. Bradshaw claims, moreover, economy for the process in regard

* *Mining Journal*, March 11th, 1893, page 276.

to time occupied in treatment, as compared with raw amalgamation in pans, stating that with an ordinary wheeler, ½ ton of auriferous stone of about 2½ ounce grade is decomposed in 6 hours, representing the capacity of one wheeler as being equal to treating 12 tons per week. Mr. McIntyre, the company's manager at Ravenswood, he goes on to state, is erecting two wheeler pans, measuring 5 feet in diameter, capable of carrying a charge of 3 tons of ore each, and it will thus be seen that with two pans, 24 tons of ore can be decomposed in 24 hours.

It would be interesting to learn how those arithmetical calculations panned out in actual practice as regards power consumed and other details.

A test of the Ravenswood ore made by Mr. Bradshaw gave an extraction of 89 per cent. of the gold and 70 per cent. of the silver.

If the success of the process demanded the mechanical incorporation of the cyanide solution with the ore in pans, it would seem likely to be foredoomed to failure, and on this account the idea is now almost entirely abandoned.

It would probably be found that, for convenience and speed, barrels could be advantageously substituted for pans, but modern developments of the process have done away with the original notion of auxiliary agitation.

The Chemistry of the Cyanide Process.

This has been ably dealt with by Messrs. Charles Butters, Ph.D., and J. E. Clennel, B.Sc., in a series of papers which appeared in the *New York Mining Journal*, in October last, from which the following extracts are taken:—

Elsner has shown that the very finely divided gold obtained by precipitating a solution of the chloride with ferrous sulphate may be dissolved by potassium cyanide, provided there is an excess of oxygen present.

The compound formed may be looked upon as a double cyanide of gold and potassium (KCyAuCy), and the reaction which takes place may therefore probably be represented by the following equation:—

$$2Au + 4KCy + O + H_2O = 2KAuCy_2 + 2KHO.$$

Two interesting points are indicated by the above equation, which it is well to bear in mind, in applying potassium cyanide as a gold solvent on a commercial scale:—

1. That the quantity of cyanide theoretically capable of dissolving a given amount of gold is infinitesimal, compared with the weight actually required in practice. Taking Au = 196·8,

K = 39·04, and Cy = 25·98, it will be observed that 130·04 parts by weight of potassium cyanide, should be capable of dissolving 196·8 parts of gold, or approximately two parts of the cyanide salt should dissolve three parts of gold. The minimum actual consumption in treating free-milling ore, assaying, let us say, 10 dwts. per ton, is about 3 lbs. per ounce of gold recovered, or roughly forty parts by weight of cyanide for one part of gold. In the leaching-tanks alone 1 lb. of cyanide is generally consumed per ton of material treated.

2. That an extremely small quantity of surplus oxygen is sufficient to cause the solution of the gold, 15·96 parts being required for 393·6 parts of gold, or say as 1 : 25. The air present in a porous mass of tailings, with that dissolved in the water used in making up the solution, is in fact more than ample to effect the reaction. To explain the enormous excessive consumption of cyanide, we must bear in mind the great instability of the simple cyanides.

Hydrocyanic acid is liberated from its salts by all mineral acids, carbonic acid, and all common organic acids. Atmospheric carbonic acid is sufficient to set up a certain amount of decomposition, in which a constant evolution of hydrocyanic acid takes place as follows:—

$$2KCy + CO_2 + H_2O = 2HCy + K_2CO_3.$$

Then, again, we must consider the proneness to oxidation which the cyanates exhibit, and which in fact lies at the base of most of their technical applications. Potassium cyanide readily changes into cyanate and ultimately into carbonate:—

$$KCN + O = KCNO;$$
$$2KCNO + 3O = K_2CO_3 + CO_2 + N_2.$$

The presence of alkalies, which always occur in commercial cyanide, tends to induce the peculiar and little understood decomposition termed hydrolysis, which seems to be mostly produced in the zinc boxes by the presence of that metal.

In the reaction alluded to above, the alkali appears to determine a chemical change in which water plays a part, while the alkali itself is not in the least affected.

There are good grounds for supposing that in dilute solutions a dissociation of the cyanide takes place, so that what we term a weak solution of potassium cyanide is in reality a mixed solution of potassium hydrate and hydrocyanic acid:—

$$H_2O + KCy = HCy + KHO.$$

This being the case, it is only natural that hydrocyanic acid should be constantly given off from all vessels in which weak cyanide solutions are freely exposed to the air, and its smell is, in fact, generally noticeable in the neighbourhood of the tanks in which it is stored.

The consumption of the reagent is on these grounds evidently enhanced by the agitation or circulation systems, since these methods involve a constant exposure of fresh surfaces.

Another source of waste is due to the tendency of the simple cyanides to form double salts with each other, or with other metallic compounds.

Salts of iron, and to a lesser extent of aluminium, magnesium, calcium, and the alkali metals are liable to occur in the tailings, especially after long exposure to atmospheric influences.

It seems, therefore, that under the most favourable circumstances an enormous waste of cyanide must take place, which may partly, however, be mitigated by the use of closed tanks and careful attention to the purity both of the cyanide itself and the water employed to dissolve it.

Action of Cyanide on Pyritic Material.—Various additional decompositions take place, when cyanide is applied to the treatment of pyritic ores or tailings. The surface ores of the celebrated banket formation of South Africa consist almost exclusively of silica and oxide of iron, the silica occurring in the form of rounded pebbles, embedded in a softer matrix highly charged with ferric oxide, which gives it a characteristic reddish tinge. The gold* is found in this matrix associated with the oxide of iron, or sometimes in small scales on the surface of the pebbles. The pebbles themselves carry little or none.

At a lower level this free-milling banket passes into an ore precisely similar in structure, but much harder, and containing the iron in the form of sulphide instead of oxide, which gives it a peculiar bluish tint.

There can be no doubt that the free-milling ores have been formed by the gradual oxidation of the pyrites through the influence of air and moisture during a long period of time, and in fact we see this change in progress, wherever pyritic material has been exposed to the action of the atmosphere. The first effect observed is the conversion of ferric sulphide

* Taking into account the production from tailings and concentrates the average yield of the Witwatersrandt ore was 12 dwts. 5 grains per ton in 1891 and 12 dwts. 13 grains in 1892. At the end of 1891 there were 1,540 stamps in operation, which yielded 11·23 dwts. per ton by direct amalgamation, whilst the average of free-gold saved fell in 1892 to 9·87 dwts., 2,035 stamps being in operation at the close of the year; the total yield having increased rather than fallen off goes to show that though the ore with depth has become more sulphuretted, it has so far been successfully treated.

into a soluble sulphate, free sulphuric acid being liberated. By the action of the air again on the ferrous sulphate, certain insoluble basic sulphates of variable and somewhat complex composition are found to result, whilst a certain amount of soluble ferric sulphate is likely to be produced at the same time.

The pyritic ores of the Witwatersrandt contain also small amounts of copper, arsenic, and sometimes cobalt and nickel, but the amount of these foreign metals has hitherto been so small that it has not practically interfered with the process.

As the fact has been observed, however, at the Robinson chlorination works, that copper and arsenic seem to occur in gradually increasing quantities with the increasing depths of the mines from which these concentrates were purchased, it is possible that these elements may be a serious source of trouble in the future.

If one attempts to treat a charge of partially oxidized pyritic tailings directly with cyanide solution, the free sulphuric acid present which renders the moisture they contain distinctly acid, sets free hydrocyanic acid. Ferrous sulphate (green vitriol) reacts upon the cyanide with the formation of ferrous cyanide, a yellowish-red flocculent precipitate :—

$$FeSO_4 + 2KCy = FeCy_2 + K_2SO_4.$$

This, however, under ordinary circumstances is slowly converted into potassium ferrocyanide by the excess of cyanide present :—

$$FeCy_2 + 4KCy = K_4FeCy_6.$$

If sufficient acid be present, the ferrocyanide reacts on an additional quantity of the ferrous salt, ultimately giving rise to a blue precipitate or coloration (Prussian blue) :—

$$3K_4FeCy_6 + 6FeSO_4 + 3O = Fe_2O_3 + 6K_2SO_4 + Fe_7Cy_{18}.$$

A coloration of that sort on the surface of the tailings or in the solution is therefore a sure indication that acid iron salts are present, and that a large waste of cyanide is taking place.

Ferric salts, when present unmixed with any ferrous compounds, decompose the cyanide solution with evolution of hydrocyanic acid, and precipitation of ferric hydrate :—

$$Fe_2(SO_4)_3 + 6KCy + 6H_2O = Fe_2(OH)_6 + 6HCy + 3K_2SO_4.$$

This reaction takes place in two stages, the first being the formation of a soluble but very unstable ferric cyanide, giving a dark brown solution :—

$$Fe_2(SO_4)_3 + 6KCy = Fe_2Cy_6 + 3K_2SO_4,$$

which decomposes as follows :—

$$Fe_2Cy_6 + 6H_2O = Fe_2(OH)_6 + 6HCy.$$

This gives rise to ferric hydrate, part of which, being in a finely divided

colloidal condition, is with difficulty removed, as it chokes the pores of the filters.

A mixture of ferrous and ferric sulphates, such as is probably always present in partially oxidized pyritic tailings, causes the appearance of a blue colour on the addition of cyanide, after the free alkali of the commercial product has been neutralized, Prussian blue (ferric-ferrocyanide), $Fe_4(FeCy_6)_3$, being produced when the ferric salt is in excess, and Turnbull's blue (ferrous-ferricyanide), $Fe_3(FeCy_6)_2$, when the ferrous salt predominates.

Before attempting to treat pyritic material or products with cyanide, it is necessary therefore to get rid of the free sulphuric acid and soluble iron compounds. This is generally done by leaching with water until the liquid running off the tanks no longer shows a coloration with ammonium sulphide. After this treatment the insoluble basic sulphates which still remain, and being gradually decomposed by water, would act upon the cyanide solution, are dealt with by washing with caustic soda or lime water. This converts the basic salts into ferric hydrate and sodium or calcium sulphates. When the quantity of free acid and iron salts is small, the preliminary wash-water may be advantageously omitted.

Lime in the dry state is sometimes mixed with the tailings before the cyanide treatment commences. When this method is adopted the iron is precipitated as a mixture of ferrous and ferric hydrates. After the washing with alkali is completed, the tanks are allowed to drain, and strong cyanide solution of about 6 per cent. is pumped on.

Even after this treatment the consumption of cyanide with moderately pyritic tailings, which have been partially decomposed by exposure, is found to be four times that which occurs with free-milling material.

The presence of a large excess of alkali in the solution brings about various secondary reactions, which lead to a loss of cyanide, such as the hydrolysis before referred to, and a peculiar action in the zinc box mentioned later.

Lime though slower in its action is preferable to caustic soda as a neutralizing agent, as it is equally effective in decomposing the iron salts, less active in producing secondary reactions on the cyanide, and also less energetic in attacking the zinc in the precipitating-boxes.

Ferric hydrate does not appear to be acted upon by potassium cyanide, but ferrous hydrate, which is formed on the neutralization of the iron salts by alkalies, reacts on the cyanide in excess, with the formation of ferrocyanide of potassium :—

$$Fe(OH)_2 + 6KCy = K_4FeCy_6 + 2KOH.$$

Deposition of Gold from Cyanide Solutions.—Under certain conditions, such as the absence of sufficient oxygen in the solution, a partial precipitation of the previously dissolved gold appears to occur. If by any chance the solution should become acid, there is a decomposition of the double cyanide of gold and potassium, in which the gold is generally supposed to be thrown down as (insoluble) aurous cyanide :—

$$KAuCy_2 + HCl = KCl + HCy + AuCy.$$

In working on the circulation-and-transfer system it is found that where pyritic material is under treatment it is not safe to transfer a solution already rich in gold to a fresh lot of tailings, as the extensive decomposition of the solution which takes place may lead to a final loss of gold.

Selective Action of Cyanide.—It is claimed by the inventors of the MacArthur-Forrest process, that in a mixture containing metallic gold, silver, copper, and base metals, cyanide of potassium exerts a selective action, dissolving first the gold, then the silver, and afterwards attacking the copper and baser metals.

The process, however, does not appear to have been successfully applied to ores, such as are met with in parts of California and Australia, containing considerable quantities of foreign metals.

Ores containing sulphides of silver and copper produce considerable decomposition of cyanide, the copper being partially dissolved as sub-sulpho-cyanide, the silver, however, remaining unattacked.

In two experiments carried out by Mr. Wm. Bettel, chief chemist of the Robinson Gold Mining Company, on an ore from the Albert silver mine (containing 30 ounces of silver per ton and 10 per cent. of copper), it was found that no extraction of silver occurred, this metal being present as sulphide.

Action of the Zinc Shavings on the Solution.—We must now consider the action of the zinc on the gold cyanide solution. Theoretically, a simple substitution of zinc for gold occurs in accordance with the following equation :—

$$2KAuCy_2 + Zn = K_2ZnCy_4 + 2Au.$$

Taking Zn = 65·1, Au = 196·8, it follows that 65·1 parts by weight of zinc should be sufficient to precipitate 393·6 parts of gold, or 1 lb. of zinc should precipitate 6 lbs. of gold. The actual consumption of zinc is about 1 lb. per troy ounce of gold recovered. It is evident, then, that zinc is consumed in some other way than in mere replacement of gold.

During the passage of the solution through the zinc boxes a constant vigorous evolution of small bubbles may be noticed, which are found to consist chiefly of hydrogen. The outflowing liquid is found to possess a

greater degree of alkalinity than it had on entering at the head of the box, and a smell of hydrocyanic acid and sometimes of ammonia is constantly observed in the neighbourhood of the boxes.

It is clear, then, that a decomposition of the potassium cyanide solution itself by the zinc is in progress, and this is not to be wondered at when we consider the powerful electro-chemical effect which must be produced by the contact of such a highly positive metal as zinc with a strongly negative metal such as gold.

Ordinary commercial zinc loses weight when immersed for some time in cyanide solution, but the action is slow. It is doubtful whether pure potassium cyanide would have any action at all on chemically pure zinc. It is well-known that the copper-zinc couple produced by immersing zinc in a solution of a copper salt decomposes water.

An analogous reaction of the zinc couple accounts for the evolution of hydrogen above-mentioned :—

$$Zn + 2H_2O = 2H + Zn(OH)_2.$$

The hydrate of zinc is at once dissolved in the excess of cyanide :—

$$Zn(OH)_2 + 4KCy = K_2ZnCy_4 + 2KOH,$$

which reaction accounts for the increased alkalinity of the solution.

There are reasons for believing that the black deposit formed on the zinc shavings is an actual chemical compound of gold and zinc, which acts as the negative element in the electric couple, the undecomposed zinc forming the positive element.

When strong solutions of caustic soda have been used for neutralizing the acid salts of the ore, a white deposit is frequently observed on the zinc. The alkali first attacks the metal to form a zinc-sodium oxide :—

$$Zn + 2NaHO = Zn(ONa)_2 + 2H.$$

This then reacts on the double cyanide of zinc and potassium always present in the solution, and precipitates the white insoluble simple cyanide of zinc :—

$$2H_2O + Zn(ONa)_2 + K_2ZnCy_4 = 2ZnCy_2 + 2NaOH + 2KOH.$$

This reaction is of some importance as affording one means by which the excessive accumulation of zinc in the solutions is avoided.

Affinity of Zinc for Cyanogen.—Potassium auro-cyanide ($KAuCy_2$) appears to be one of the most stable of the salts of gold, and the reaction in the zinc boxes, shows that the affinity of zinc, together with potassium, for cyanogen, is greater than that of gold with potassium for the same radicle. Hence a solution of potassium cyanide cannot dissolve gold which is in contact with zinc, neither can gold replace zinc in a solution of the double cyanide of zinc and potassium. So long as any zinc is present,

therefore, we need not fear that the precipitated gold will redissolve in the excess of potassium cyanide flowing through the boxes.

It is evident also that the cyanogen contained in the double cyanide of zinc and potassium is not available for dissolving gold, nor when a solution charged with zinc is employed in the treatment of a fresh lot of tailings it is only effective in so far as it contains a certain quantity of simple cyanide of potassium or other alkaline cyanide.

New Methods of Precipitation.—The cyanides of sodium and ammonium and those of the alkaline earths (calcium, barium, etc.) will dissolve gold as well as potassium cyanide. Sodium cyanide is more difficult to manufacture than the potassium compound, but a given weight of it should be more effective than the same weight of potassium cyanide, inasmuch as 49 parts of the former are equivalent to 65 of the latter.

The advantages of Molloy's process and others which employ sodium or potassium amalgam will be referred to later. The alkali metal is obtained by the electrolysis of the carbonate between electrodes of lead and mercury :—

$$Na_2CO_3 = Na_2 + CO_2 + O.$$

The sodium forms an amalgam with the mercury. Sodium amalgam may also be manufactured direct from its elements. It is claimed for this method of precipitation that the whole of the cyanogen is restored to a condition available for dissolving gold as shown by the reaction :—

$$Na + KAuCy_2 = Au + KCy + NaCy.$$

Composition of the Zinc Slimes.—Any base metals which happen to be in solution in the cyanide liquor are liable to be precipitated by the zinc along with the gold. Hence the zinc slimes are found to contain a certain percentage of copper as well as traces of arsenic and antimony. Moreover, any impurities in the zinc will also find their way into the slimes, as zinc will be dissolved by the cyanide in preference to any less oxidizable metals, *e.g.*, tin and lead. Silver is dissolved by the cyanide and re-precipitated by zinc by a set of reactions precisely analogous to those of gold :—

$$2Ag + 4KCy + O + H_2O = 2KAgCy_2 + 2KOH, \text{ and}$$

$$2KAgCy_2 + Zn = K_2ZnCy_4 + 2Ag.$$

It has been observed that the proportion of silver to gold is greater in the cyanide bullion than in the gold from the batteries, and this is explained by supposing that the loss of silver in amalgamation is greater than that of gold.

Treatment of the Zinc Slimes.—The removal of the zinc is a troublesome operation, and is only very partially carried out in smelting the

dried slimes. The admixture of sand is made for the purpose of forming a fusible silicate of zinc.

A portion of the zinc is volatilized and burns at the mouth of the crucible with a greenish flame, producing the white oxide (ZnO), which is found encrusting the flues, and no doubt carries with it no inconsiderable quantity of gold and silver. The most promising method of treating these slimes appears to be that suggested by Mr. Bettel of fluxing with acid sulphate of soda and fluorspar.

Attempts to remove the zinc prior to smelting have been only partially successful, as all such methods involve the filtration of a slimy mass, which retains soluble salts with great tenacity.

The slags from the fusion of the zinc slimes contain a considerable amount of gold, some of which is in the form of round shot, and may be removed by pounding up the slag, passing through a coarse sieve, and panning off. The residue from the first fusion should always be fused again with additional lead to form an alloy with the gold. The same lead bars may be used for a number of successive fusions of the slag, and when sufficiently enriched the gold can be recovered by cupellation.

Testing of Cyanide Solutions.—It is a matter of importance to determine exactly what strength of cyanide solution is used in treatment of tailings. The ordinary method of testing depends on the fact that silver cyanide is soluble in excess of potassium cyanide, with the formation of a double cyanide of silver and potassium :—

$$KCy + AgNO_3 = AgCy + KNO_3, \text{ and}$$
$$AgCy + KCy = KAgCy_2.$$

When silver nitrate solution is added drop by drop from a burette to a solution of cyanide a white precipitate is formed, which quickly redissolves. At a certain stage the precipitate becomes permanent, when, in fact, the whole of the cyanide has been converted into the soluble silver salt, and an additional drop of silver nitrate produces a permanent precipitate of the insoluble simple cyanide of silver :—

$$KAgCy_2 + AgNO_3 = KNO_3 + 2AgCy.$$

From these reactions 107·66 parts by weight of silver are equivalent to 130·04 parts of potassium cyanide. A convenient standard silver solution is one of such strength that every cubic centimetre added to 10 cubic centimetres of the solution to be tested, corresponds to 0·1 per cent. of pure potassium cyanide.

This method gives good results when pure cyanide solutions are under examination, but when the solutions contain zinc it is difficult if not impossible to determine the end of the reaction. A white flocculent pre-

cipitate occurs at a certain stage, probably consisting of simple (insoluble) cyanide of zinc, formed by the decomposition of the soluble double cyanide.

This precipitation occurs long before the whole amount of potassium cyanide has been converted into the soluble double salt of silver ($KAgCy_2$) for the solution after the appearance of the flocculent precipitate still gives the Prussian blue reaction with acidulated ferrous sulphate.

A standard solution of iodine in potassium iodide may be used with great accuracy for determining the total amount of cyanogen in a solution whether in combination with zinc or not, making use of the reaction :—

$$KCy + I_2 = KI + ICy.$$

The colour of the iodine is discharged so long as an excess of cyanide is present. The sharpness of the terminal reaction may be increased by adding a small quantity of starch to the solution under examination, which gives a permanent blue colour as soon as an excess of iodine has been added.

What is most needed, however, is a rapid method of determining the amount of cyanide available for dissolving gold, for, as was pointed out above, the cyanide in combination with zinc is not available for that purpose.

The method of testing solutions containing zinc for available cyanide which was introduced by Mr. Bettel at the Robinson Company's works is as follows :—Two perfectly clean flasks of equal size are taken. To each of these is added a considerable bulk, say 50 cubic centimetres of the solution to be tested and 50 cubic centimetres of water. The liquid in both flasks will probably appear slightly turbid, but the degree of turbidity will be the same in each. Standard silver nitrate solution is run into one flask until the slightest possible increase in turbidity is observed in comparison with the liquid in the other flask. This point is taken as indicating the conversion of the whole of the free potassium cyanide into the soluble silver salt, and therefore as determining the amount of available cyanide present.

The amount of gold in the solution is generally found by evaporating a known bulk with litharge, fluxing the residue and cupelling the resulting lead bullion. Evaporation on lead foil may likewise be employed.

Poisonous Properties of Cyanide.—A few words may not be out of place in this connexion. Although one of the most rapidly and deadly of known poisons when taken internally, its action as a blood poison is much less violent. Nevertheless, when introduced into cuts it produces very painful sores. The men employed in cleaning-up and in melting the slimes are subject to a peculiar eruption, especially on the arms, and complain of headache, giddiness, and general depression.

Ferrocyanide of potassium has been recommended as a remedy for the eruption; it may be taken internally and also applied as a lotion. Considering the dangerous nature of the reagent it is remarkable how few fatal cases have occurred through the use of cyanide on a large scale. In cases of poisoning precipitated carbonate of iron, obtained by mixing solutions of sodium carbonate and ferrous sulphate, may be used as an antidote, forming internally an insoluble blue compound with the cyanide.

Hydrocyanic acid acts directly on the nervous system, causing instant paralysis, hence any treatment which will excite the action of the nerves such as applications of cold water to the spine, inhalation of ammonia, etc., may be tried in cases of faintness produced by breathing the acid vapour.

The disposal of waste cyanide liquors is a matter of serious consideration. Solutions containing 0·1 to 0·2 per cent. of potassium cyanide must occasionally be discharged and are likely to contaminate the water of the dams or streams which receive them to a dangerous extent. If some effective means of precipitating the zinc, or better still of dispensing with the use of zinc altogether, could be devised there would never be any necessity for allowing cyanide liquors to leave the works.

Applications of the Cyanide Process.

Mr. Almarin B. Paul appears to have originated the idea of using cyanide solution in the battery crushing the ore wet. This he claims to have done with success at the Calamut mill, Shasta County, California. He states that his plan is in all cases to crush with a weak solution, and should the ore require a higher percentage of cyanide when the first solution has percolated below the surface of the ore, after the tanks have been filled, a stronger one can, if necessary, be introduced. All the dust and disagreeable effects of dry-crushing is thus avoided. The loss of cyanide in crushing is but nominal, and is off-set by the cheapness of working and completeness of distribution of the cyanide through the pulp in the tanks. This plan has been followed of late at the May Consolidated works in South Africa, but it does not appear to be applicable to all cases, as generally when ores have been crushed wet and run direct to the tanks the pulp has packed so hard as to be impenetrable to the leach liquor. Ores containing impurities, such as arsenic and tellurium, have proved obstinate to deal with.

The consumption of available potassium cyanide may for practical purposes be divided into avoidable and unavoidable decomposition; each of these factors varying with the composition of the ore, the strength of the solution, time of contact, and method of treatment.

Tests made on a heavy, raw sulphide gold ore by Mr. C. W. Merrill, using a 2 per cent. solution, showed that it was capable of decomposing available cyanide at the rate of 12 lbs. per ton in 24 hours, but that by taking all possible precautions this loss could be reduced to 3 lbs. per 24 hours.

The reduction in loss of cyanide seems, however, to have been set off by an extravagant loss of time, as it is necessary to continue the treatment for 7 to 10 days.

A series of extraction tests proved that a $\frac{1}{4}$ per cent. solution when reinforced each day gave as good results as a stronger one, and with a decomposition of only 5 to 7 lbs. of cyanide per ton. As the ore contained acid salts of iron and fine copper sulphides, both of which decompose potassium cyanide, the probability is that the weak solutions were rendered inert after a few hours' contact with the ore, hence the necessity of reinforcing them.

A series of experiments, published in detail in the *New York Mining Journal* of December 24th, 1892, were made by Mr. G. E. Kedzie, M.E., to determine how far the gold-bearing pyritic ores of Ouray, Colorado, could be successfully treated by the cyanide process, and also to learn the conditions under which the most complete extraction could be obtained, together with the amount of cyanide consumed. These experiments throw much light on the question of how far it is practicable to treat ores of a similar description by the cyanide method.

The samples under treatment contained gold and silver in varying proportions, ranging from 0·27 ounce to 145·90 ounces of the former, and 0·65 ounce to 458 ounces of the latter metal, the value of the high-grade ores and tailings being determined by triplicate scorification assays, and that of the low-grade ores and tailings by duplicate crucible fusions. The gangue in some cases was quartzose, in others calcareous or clayey, and contained iron pyrites, copper pyrites, and magnetite, either alone or in admixture.

The deductions drawn from these tests were: (1) the gold is more readily extracted than the silver; (2) under the same conditions the percentage of extraction is increased—(*a*) by the fineness of the pulp—(*b*) by the duration of treatment—(*c*) by the strength of the cyanide solution; (3) the greater the amount of cyanide added to the ore, the higher will be the percentage of extraction, but in this case the total values extracted for each pound of cyanide consumed are less than when a smaller amount of cyanide is used for each ton of ore treated; and (4) when the same amount of cyanide is used for each ton of ore treated the percentage of extraction is greater when the weight of the solution is equal to that of the ore taken.

His conclusions are that there are no flattering indications of the process being a metallurgical success with the pyritic ores under consideration. With high-grade ores, which are under no circumstances adapted to this process, the percentage of extraction under the most favourable circumstances is low, lower even than the results obtained by amalgamation. However, the total values extracted for each pound of cyanide consumed are relatively high.

With low-grade ores, even where the low value of the tailings will admit of their being thrown away, the total values extracted in a majority of instances are less than the cost of cyanide consumed, to say nothing of milling expenses. In these experiments the percentage of potassium cyanide varied from 0·5 to 1·5 per cent. The treatment lasted from 12 to 60 hours. The extraction of the gold ranged from 2·31 to 84·62 per cent. The extraction of the silver varies from *nil* to 88·88 per cent. The proportion of the potassium cyanide added per ton of ore represented from 5 to 60 lbs., and its consumption rose from 3·2 to 50·8 lbs. per ton. The lowest value extracted per lb. of cyanide consumed was 0·28 dollar and the highest 53·72 dollars. The latter result was obtained in treating an ore which assayed 145·90 ounces in gold and 458 ounces in silver per ton.

Mr. Kedzie suggests, as the outcome of his observations, that the strongest solutions used should not exceed in strength $\frac{1}{2}$ per cent. of potassium cyanide, and good results are anticipated with much weaker ones. Arrangements should be made for the slow percolation of the cyanide solution, so as to keep the ore always in contact with a solution of nearly normal strength, and finally the period of lixiviation should be extended to 96 hours, if necessary, in order to secure a more complete extraction. To comply with these conditions the mechanical difficulties of leaching and loss of time must evidently be debited against a higher return.

Messrs. Clennel and Butlers state* that the cyanide process is the only method which has hitherto met with success in treating the tailings derived from various South African mills after amalgamation. The adoption of the process has been rapid, and its success most remarkable. Despite its simplicity on paper, the working metallurgist who attempts to carry out the process on a large scale soon finds himself confronted with difficulties. These arise from the nature of the material under treatment, and from the manipulations necessary in applying the treatment to ore in bulk.

Neither the solution of the gold in the ore under treatment, nor its precipitation, nor the conversion of this precipitate into bullion, is perfect

* Continuation of articles in the *New York Mining Journal* before referred to.

theoretically or practically. Losses occur in each operation, and the consumption of cyanide and zinc has been shown to be much in excess of what is indicated by the various chemical reactions primarily involved.

Hitherto the process has only been successfully applied to those ores or tailings usually described as free-milling, *i.e.*, such as are capable of yielding the greater part of their gold in the ordinary amalgamation process. The promoters of the process have been singularly fortunate in South Africa in the nature of the material they have had to deal with. The ores forming the upper portion of the main reef series, extending to a depth of 10 to 150 or 200 feet (as on the Robinson property) consist almost exclusively of silica and oxide of iron. They contain practically no substance, except gold and silver, which the cyanide is capable of attacking.

It is in dealing with ores or products containing sulphide of iron, especially when these are partially oxidized to sulphates and those containing compounds of lead, zinc, etc., that the difficulties of the process begin.

Solution of the Gold.—When the cyanide process was first introduced, about two years ago, it was thought necessary to agitate the material under treatment with the cyanide solution to obtain a satisfactory extraction. It was soon found, however, that the power needed and the rapid decomposition of the reagent were items of expense, which more than compensated for the greater percentage of the precious metals obtained in comparison with simple lixiviation. The operation is carried out by the African Gold Recovery Company, who represent the patentees in South Africa, as follows :—The damp tailings from the settling-pits are charged into wooden vats holding 35 to 50 tons, originally built square. The best works are now building them circular, those at the Robinson works holding 75 tons, and at the Langlaagte Estate 400 tons, whilst at the New Primrose still larger ones are being constructed.

The vats are filled to within a few inches of the top, and the surface of the tailings levelled. Cyanide solution of 0·6 to 0·8 per cent. strength is then allowed to flow into the tank until it is entirely filled.

The ore settles from 3 inches to 1 foot below the rim of the tank (the amount of shrinkage depending on the depth of the vat). This solution is allowed to remain undisturbed in contact with the ore for 12 hours.

Each vat is provided with a false bottom (usually a wooden framework covered with cocoanut matting). Below this is a layer of coarse sand and pebbles through which the solution percolates,

An iron pipe communicates with the vat below the false bottom, and conveys the filtered solution to the zinc boxes, where the precipitation takes place. After 12 hours' contact with the ore, the solution is allowed to drain out of the tank by opening a cock.

The dilute cyanide solution does not attack the wood, or corrode the iron piping to any appreciable extent. In wear and tear of the apparatus the use of cyanide, therefore, offers advantages over chlorine, which otherwise might be applied just as cheaply to the treatment of these oxidized surface-ores. Brass plungers and pump-valves are attacked, but not very rapidly. The pumps at the Robinson works stood for four months with comparatively little wear, but iron is preferable for the pump fittings where cyanide is employed.

As the liquor is drawn off during the leaching process it is replaced by fresh solution. This operation is continued for a longer or shorter period, depending on the grade of the tailings, for 6 to 12 hours. At the end of this period, known as the strong solution-leaching, a weaker solution (containing 0·2 to 0·4 per cent. of cyanide) is turned on, and allowed to filter through the ore for about 8 to 10 hours. This weak solution is then drawn off through another zinc box (known as the weak zinc box). Finally, a quantity of water is run into the tank, more or less equivalent to the amount of moisture which the ore contained when the tank was filled. This last washing-water displaces the weak cyanide solution, so that the volume of cyanide solution in use remains unchanged. The weak solution is in fact the liquor which has previously passed through the zinc boxes into the receiving-tanks or sumps, and has been pumped again to the leaching-tanks. The cyanide is usually supplied in cases containing 170 to 195 lbs. of crude cyanide, contaminated with carbonaceous matter and iron, but containing 72 to 78 per cent. of pure potassium cyanide.

This cyanide is usually dissolved in a small volume of water to form a highly concentrated solution, a special tank being employed for this purpose. The solution of required strength is obtained by adding this concentrated solution to the dilute liquor in the sumps. By this means a dilute solution of a given strength is more accurately arrived at than if the cakes of cyanide were dissolved directly in the required volume of water, since the percentage of cyanide can be more accurately determined in a strong solution. The actual amount of solution used is about half a ton of the stronger (0·6 to 0·8 per cent.) and half a ton of the weaker (0·2 to 0·4 per cent.) solution for every ton of ore leached.

When the final wash-water has been added, and has displaced the weak solution, the residues are discharged usually by the tedious process

of shovelling them over the side of the vat. A truck line runs generally by the side of the tanks to receive the discharged tailings, which are dumped outside the works. The tank is then ready for a fresh charge.

The percolation system just described may be modified however in various ways. One of the first difficulties arose when the tailings contained a small percentage of iron pyrites, which, on exposure to the air, had become converted into sulphate, with the liberation of free sulphuric acid. The trouble could be minimized by treating the tailings direct from the battery, giving no time for the oxidation of the pyrites, but to deal with old tailings which had been exposed to the atmosphere for some time it was found necessary to give them a preliminary washing, first with water and afterwards with an alkaline solution, such as lime or caustic soda. At the Robinson works lime was given the preference, as it is less active in inducing decomposition of the cyanide solution in the tanks and in attacking the zinc used in precipitation.

Substances present in the ore which are capable of decomposing the cyanide may cause much trouble by reprecipitating the gold, and depositing it in the gelatinous mass formed on the surface of the tank, especially if the circulation-method is employed. It has been stated that a ton of ore generally requires for its treatment a ton of solution, and since with free-milling ore a much smaller quantity is sufficient to dissolve the gold, it was suggested that the solution from one tank might be transferred to a second, and be made to dissolve an additional quantity before passing to the zinc boxes. For example, at the Robinson works it was found that 20 tons of solution were amply sufficient to extract 40 ounces of gold from 75 tons of tailings in one tank; whilst 20 tons of solution sufficed to fill a tank holding the usual charge of 75 tons of tailings, covering it to a depth of 3 or 4 inches.

Instead, therefore, of replacing this 20 tons of solution by fresh cyanide, the solution filtering through was continually pumped back again into the same tank for about 36 hours, and then passed through the zinc box.

The extraction of gold by this circulation-system was equal to that obtained by the ordinary method, and the consumption of cyanide was much less, since a much smaller quantity of solution was exposed to the zinc. Another modification suggested itself as a further step, viz., to transfer the solution charged with gold from one vat to a second and third in order that it might take up an additional quantity of gold from fresh tailings before going through the next stage of the process.

The advantages of this method are that the solutions from which the precipitate is obtained are much richer in gold, giving a cleaner deposit on the zinc with much less consumption of cyanide.

Whilst the usual practice is to prepare the cyanide solution in a special tank, as before stated, at the Robinson works, the strength of the liquor is kept up by adding cyanide in lumps, dissolving it under the stream from the pump. By this procedure, a cyanide solution, of required strength, is formed in the leaching-tank itself.

This simplifies the operation and diminishes the number of tanks needed, while affording an easy means of getting rid of the insoluble impurities of the cyanide (the so-called carbide of iron) which would otherwise accumulate as a black slimy deposit in the concentrated solution tank. This insoluble residue is, of course, discharged with the tailings when the tanks are emptied.

A further difficulty frequently encountered in the application of the cyanide process is the treatment of battery slimes, which, owing to their fine state of division have a tendency to agglomerate into pasty masses. These either resist the penetrating action of the cyanide or retain the dissolved gold during the leaching operation.

No satisfactory remedy had at any rate until quite recently been devised, though the evil may be lessened by mixing the slimes with clean, coarse sand. In the direct treatment of ore from the battery previously alluded to, coarse gold, which is easily caught on the plates, is very slowly dissolved by cyanide, and this is probably a serious source of loss.

The experience at the Langlaagte Estate and other works in treating pyritic concentrates with or without agitation seems to confirm what has been already stated. Although an extraction of over 90 per cent. has been obtained, it appears that the consumption of cyanide is enormous.

Various improvements have been introduced in the mechanical details of construction. The large leaching-vats of the Robinson works are rapidly and conveniently discharged by a trap-door placed in the centre of the tank bottom, which is hermetically closed by a screw-fastening. The enormous underground vats of the Langlaagte Estate works are discharged by a dredge, which appears to give satisfaction.

Precipitation of the Gold.—Having obtained a solution of gold in cyanide of potassium, the next step is to recover it. Various precipitants have been suggested, but the only one which has come into use on a large scale is metallic zinc in the form of freshly-turned shavings; zinc in sheets offer too little surface for the deposition. The same is true of granulated zinc. When once the surface has become coated with an extremely fine layer of gold the action ceases, or becomes so slow that the precipitant cannot be practically applied in this form. Zinc dust and zinc amalgam have also been tried and are effective, in so far as they

present a large surface for deposition, but are found to clog if the continued flow of liquid through them be interrupted. Sodium or potassium amalgam has been used with success on a small scale, as in the Molloy process now about to be tried with a 500 ton plant at the works of the Pioneer Gold Mining Company, but the difficulty so far has been to manufacture these substances cheaply and in sufficient quantity on the spot.

The zinc shavings now in use are prepared by turning thin sheets of zinc on the lathe. This produces a light spongy mass, which readily allows the solution to filter through and presents a large surface for the precipitation of the gold. These shavings are placed in wooden troughs, commonly known as zinc boxes, and the solution from the leaching-vats is allowed to flow slowly through them, depositing the gold as a finely divided black slime on the surface of the zinc, while the zinc gradually dissolves in the liquid. After passing the zinc box the exhausted solution, which should not contain more than $\frac{1}{2}$ dwt. of gold per ton, flows into the storage tank or sump, whence it may be pumped back to the leaching-tanks, when a fresh charge has to be treated.

The simple replacement of gold by zinc is not the only reaction which occurs in the zinc box; we find that a notable falling off in the strength of the cyanide occurs, due to secondary reactions caused by the gold-zinc couple.

Various slight modifications have been introduced in the construction of the zinc boxes. They are usually divided into several compartments so arranged that the liquid flows alternately upward and downward through the shavings. The shavings are placed in a tray, the bottom of which is an iron-wire screen of about 4 holes to the inch. This is supported a few inches from the bottom of the zinc box. The fine gold slimes fall through this screen, and may thus be separated from the undecomposed zinc when the clean-up takes place. The zinc boxes used at the Robinson works are about 20 feet long, 2 feet wide, and 2 feet deep, with inclined bottoms. They are divided into compartments of about 20 inches in length. Each compartment holds about a bushel of shavings, weighing about 40 lbs.

Seven compartments in each zinc box are filled with shavings; a single compartment at the head is left empty to receive any sand that may be carried through the filters by the solution from the tanks. A double compartment at the foot is also left empty to allow any gold that may be carried away by the stream of liquid to deposit before the solution flows into the sump. About 60 tons of solution, which is the quantity required for treating the ordinary daily charge of 225 tons of tailings, are allowed to run off through two zinc boxes in about 9 hours. This solution may

carry from 1 to 3 ounces of gold per ton of liquid; after passing through the zinc boxes it rarely contains more than 2 dwts., and should not contain more than ½ dwt. if the precipitation has been properly carried out.

There are two sets of zinc boxes, one to receive the strong solutions (0·6 per cent. to 0·8 per cent. cyanide), and one for the weak solutions (0·2 per cent. to 0·4 per cent.). The slimes formed in the weak boxes are, as a rule, much poorer than those in the strong boxes, and less consumption of zinc takes place in them.

The total amount of zinc consumed amounts to about 100 lbs. per day. Two men are constantly employed at the lathes, so that the turning is an arduous and somewhat costly operation. It is desirable to use freshly-turned zinc, as the surface rapidly oxidizes and then becomes much less active in precipitating the gold. The most vigorous action of course takes place in the compartments which first receive the solution from the tanks. It is here that the zinc dissolves most rapidly, and is accordingly replaced by shavings from the lower compartments, whilst fresh zinc is continually added as the last compartment is emptied.

The clean-up takes place once or twice a month. The screens containing the undissolved shavings are lifted from the zinc boxes. The boxes are then left undisturbed for an hour or so to allow the zinc gold slime to settle at the bottom. The liquid is then drawn off by a syphon until very little is left above the slimes. The box is then cleaned out, and the slimes and muddy water allowed to drain through a screen of 40 meshes to the inch. The mass consisting of water, finely divided gold, and very fine zinc is rubbed through this screen with a short stick 5 or 6 inches long, to the end of which a piece of india-rubber is fixed.

The stuff remaining on the screen consists almost entirely of unconsumed zinc fine enough to pass through a screen of 12 meshes to the linear inch. This is replaced in the first division of the zinc boxes over a fresh lot of shavings. The slime consisting of finely divided gold and silver, with a large proportion of zinc and lead, and a certain quantity of tin, antimony, organic matter, and other accidental impurities, is allowed to settle in a small tank placed beneath the 40 mesh screen, and is now ready to undergo the drying and melting operations necessary to convert it into bullion.

In the Molloy process the use of zinc is dispensed with altogether. The solution passes through a shallow trough containing mercury, in which is an inner cylindrical vessel filled with solution of carbonate of soda, the edges of the cylinder just dipping beneath the mercury so that its contents are entirely cut off from the other portion of the vessel.

A rod of lead dips into the soda solution, the lead and mercury are connected with opposite poles of the dynamo and the solution is electrolysed by the passage of the current. The sodium released combines with the mercury to form sodium amalgam, which decomposes the gold cyanide solution with formation of gold amalgam, sodium cyanide being simultaneously produced.

It is claimed that much less decomposition of the cyanide takes place than with zinc, and that the outflowing solution is better adapted for dissolving fresh quantities of gold, becoming regenerated. In the ordinary method a large accumulation of zinc in the solutions must take place, which in time would render them valueless for gold extraction, whereas sodium cyanide is just as effective as the potassium compound. If this method of precipitation should prove successful on a large scale, a great improvement will have been effected.

Production of Bullion.—The third stage of the cyanide process consists in converting the precipitated gold into a saleable form. The slimes are transferred to enamelled iron pans and carefully dried over a small furnace. This is a tedious operation which requires time. The richness of the dried slimes will depend on the percentage of gold present in the cyanide liquors passed through the zinc boxes.

The pans in use at the Robinson works contain about 5 or 6 gallons of dried precipitate. This may contain as much as 150 or as little as 20 ounces of gold. The precipitate when nearly dry is mixed with sand, borax, and bicarbonate of soda, and melted in a No. 60 crucible at a fairly high temperature. The material melts very easily, forming a very liquid slag which, however, corrodes the pots rapidly, so that a good crucible rarely lasts out more than eight meltings. The change is not added all at once, but as each portion melts and sinks down fresh quantities of the mixture are added.

When the pot is full of liquid slag it may contain from 100 to 150 ounces of bullion. Large quantities of oxide of zinc are given off during the melting, which carry off a very appreciable amount of gold. The zinc fumes, together with the products formed by the decomposition of the cyanide salts, render the operation anything but healthy. The bullion produced is whitish in appearance and about 650 fine. It is very hard and brittle, and the bars are by no means uniform, so that it is difficult to obtain an accurate assay. In addition to zinc they contain silver, lead, and sometimes a little copper. Several ways have been suggested for obtaining a purer bullion. One method consists in partially roasting the slimes in a muffle furnace, whereby part of the zinc is oxidized and

volatilized, leaving a much smaller mass for subsequent melting. By this treatment bullion of 800 fine may be obtained.

Another method is to carefully wash out the soluble cyanide salts and then treat them with dilute sulphuric acid, which dissolves the zinc. The objection to this process is that the slimy mass is very difficult to filter, and retains the soluble zinc salts even after prolonged washing.

The same trouble occurs when the zinc is dissolved in hydrochloric acid. Some difficulty is also experienced in washing out the soluble cyanide salts. It is possible that the use of filter-presses might to some extent solve the difficulty of purifying the zinc slimes. The use of acid sodium sulphate also appears to promise good results as a solvent for zinc.

No attempt as a rule is made to refine the product, because to melt straight away into bars of bullion yields a larger number of ounces per month, though it does not augment the sterling value of the product.

In general it is found that on a large scale the extraction of gold by cyanide amounts to 70 or 80 per cent., and tailings assaying 8 to 10 dwts. will give residues assaying 2 to 2½ dwts.

A much higher extraction is obtained on a small scale with the same strength of solution when, relatively, large quantities of solution are allowed to pass through the material under treatment.

That the cyanide process is well adapted at present for the treatment of Witwatersrandt ores is shown by the fact that, although the system has been in use little more than two years, 40,000 tons of tailings are now being treated per month. The process is only in its infancy, the varied and complex problems to which it has given rise are mostly unsolved, its limits are yet undefined, and it would be rash indeed to forecast its future.

Turning again to what has been done in America, some interesting particulars of the process are given by Mr. C. Merrill in the *New York Mining Journal* of November 5th, 1892, from a report made by Messrs. Louis Janin, Sen., and Henry Bratnober on the results of the process at the Mercur gold-mine, Fairfield, Utah. The ore is a siliceous limestone, carrying magnetic oxide of iron, traces of cinnabar, and gold (no silver). No free-gold appears to be present—in the ordinary acceptation of the term—though the magnetic oxide of iron appears to be more or less coated with a thin film of gold. The ore contains considerable silt, and any attempt at fine-crushing results in sliming the greater portion. This being fatal to successful leaching, coarse-crushing had to be resorted to, which has the disadvantage of involving more time for successful treatment. There appear in fact to be few ores which can be treated with any

degree of success unless crushed fine enough to pass a 20 or 30 mesh screen. The ore after crushing in a stone breaker passes through two sets of corrugated rolls, which give a product, 20 per cent. of which remains on a No. 4 screen, 40 per cent. on a No. 12, 13 per cent. on a No. 30, and 26 per cent. passes through. Of this last 26 per cent. nearly the whole is impalpable powder.

The ore is carried by an overhead tramway on cars to the leaching-vats. The best size and pattern for the latter, according to experience at this mill, is a round vat, the shell of which is No. 10 or 12 sheet-iron. The bottom is of 3 inch California red wood, and is caulked with oakum, over which is poured a mixture of tallow and resin. On this bottom are placed 1 inch by 1 inch slabs 18 inches apart. Upon this rests a false-bottom of 1 inch yellow pine, in every square inch of which is perforated a $\frac{3}{4}$ inch auger hole. Over this perforated false-bottom is stretched a burlap filter. There is, of course, a stopcock (between the true and false-bottom), which should be of iron. The dimensions of the vat are: diameter, 12 feet 8 inches; depth over all, 40 inches; depth to false-bottom, 35 inches; capacity, 14 tons. There seems no reason, however, why larger vats should not be used, provided they are round.

The ore having been charged, is levelled to within about 6 inches of the top of the vat, and the stopcock being closed, a $\frac{1}{4}$ per cent. solution (1 lb. of pure potassium cyanide to 400 lbs. of water) from the standardizing-tank, is run in from a pipe till about 3 inches of solution covers the top of the ore. It has been found that the solution acts slowly at first, but more rapidly after extraction has begun, possibly owing to some galvanic action.

The charge is allowed to soak from 12 to 24 hours, then the solution is allowed to percolate, flowing in at the top and out at the bottom for from 24 to 240 hours, according to the leaching rate of the ore. That from near the surface of the mine is very slimy and requires a longer time. The average time is about 60 hours, or practically until the outgoing solution does not discolour bright zinc. The test is made by placing a little sieve containing bright zinc threads under the stopcock, and allowing it to remain there for an hour or so.

If it remains bright the solution has extracted all that it will extract, and the flow is stopped. The outgoing solution from all the vats flows to a sump, and is pumped thence to a second or gold-solution tank. From this tank it is allowed to flow constantly through two boxes containing spongy or thread zinc, each box being 40 feet long, one of wood 12 inches square internally, and the other of iron 15 inches square. Each box is provided

with partial partitions which deflect the current from the bottom to the top and *vice versa*. These partitions are placed about 3 feet apart. The zinc should occupy so many of them as will give bright zinc in the last one, so as to ensure complete precipitation. The solution flows from the zinc boxes back to the standardizing-tank, where it is occasionally tested, and if below the required strength, potassium cyanide is added in proper quantities.

After the gold has been extracted and the solution has been turned off, the vats are allowed to drain. It has been found that there remains 400 lbs. of $\frac{1}{4}$ per cent. cyanide to the ton in the tailings, and to force this out a wash of about the same quantity (400 lbs.) of water is used, which joins the main body of rich solution in the sump.

After this is drained out, as the tailings still contain 0·3 to 0·4 of a pound of cyanide per ton, a second wash of 400 lbs. of water to the ton is added, and the very weak solution left (which is forced out), runs to the waste-solution tank, whence the first wash of succeeding charges is drawn.

At the end of the month the outlet from the gold-solution tanks to the zinc boxes is closed and the latter are allowed to drain. When comparatively free from solution the richest portion of the zinc product, which has the property of powdering up in the fingers, is removed.

Mr. A. Hanauer gives the following analysis of this zinc and gold product:—Zinc, 39·1; carbonate of lime, 36·7; gold, 4·4; cyanogen, 3·5; sulphur, 2·6; iron, 2·4; and residue, 6·0. This product is sampled and treated by the Omaha Smelting Company, who return 20·60 dollars per ounce of their assay of its gold contents. Deducting 12 cents for express charges, this leaves a net return of 19·88 dollars per ounce of gold extracted.

It is stated that from April 1st to July 1st, there were milled 1,513 tons of moist ore of an average value per dry ton by assay of £3 3s. 5d. (15·22 dollars). The ore tailings' assay was 10s. 10d. (2·60 dollars). The apparent extraction was therefore £2 12s. 7d. (12·62 dollars) per dry ton, or 83 per cent.; hence, without allowing for moisture, we should have a bullion return of 19,272·60 dollars. The actual bullion returns from the smelter were 16,805·80 dollars = 73 per cent. = £2 6s. 0d. (11·04 dollars) per ton of moist ore. The discrepancy is accounted for by the ore being weighed wet and assayed dry, making a difference of at least 6 per cent. due to moisture, besides loss by leakage, handling, and drying.

The losses due to leakage and handling will probably be minimized when the mill is in thorough working order, and should, it is claimed, leave a difference of not more than 2 to 3 per cent. The following is

stated to be the itemized cost per ton, compiled from the books of the company. It is exclusive of superintendence, office expenses, and royalty, and covers a period of six months from January 1st to July 1st :—

	s.	d.
Potassium cyanide, 1·27 lbs. per ton	2	9
Zinc, 0·55 lb. per ton	0	2½
Labour (7 shifts per 24 hours, 6 day and 1 night) ...	4	8
Supplies, repairs, fuel, freight, etc.	2	4½
	10	0

In regard to the labour item it is said that since the period covered by these figures the capacity of the mill has been doubled, reducing this charge to 2s. 4d. per ton, and the total cost to 7s. 8d. per ton. It is intended in the near future to again double the capacity of the works, requiring only the addition of 4 shifts to run the crushing machinery and to charge the tanks at night. At present only the solution-man is on night shift. Notwithstanding the difficulty the Mercur ore offers to leaching, (owing to its disposition to slime), the process appears to be in this case certainly an economic success, but it cannot be repeated too strongly that extensive preliminary investigations should be made by an impartial person before attempting to apply it to an untried ore in a new locality.

Raw Grinding.

As to the amalgamation of gold ores, Messrs. McDermott and Duffield remark :* "Generally the pan-amalgamation process must be preceded by roasting. In the colonies, grinding and working in pans the raw, partially concentrated sulphides is practised, but in the United States this process has entirely gone out of use, being displaced by smelting and chlorination, rendered possible by the development of the country and more perfect concentration."

Some sulphides contain their value in comparatively coarse-gold, which, grinding will liberate and mercury attack, but most of the base-metal sulphides in gold ores will only yield a small part of their gold contents to such a process. Even when still in use this process has its chief excuse in an imperfect concentration, and would be better superseded by the close saving of higher-grade clean concentrates, properly treated by chlorination, or some modification of the smelting process.

The same writers remark :† "The sulphide ores and concentrates can, as a safe general rule, be considered as not adapted to amalgamation without a previous roasting. In some few cases where the gold is comparatively coarse, a sufficient proportion of it can be extracted by

* *Gold Amalgamation*, page 31. † *Ibid.*, page 44.

raw pan-amalgamation to pay a profit on the operation, and this process is used at places in the Australian colonies, usually after a crude process of concentration, which itself involves considerable losses of fine minerals, and the production of unclean concentrates, and in some few cases the value thus extracted is to an appreciable extent in the form of amalgam lost before concentration. Generally speaking, therefore, it is safe to say that raw amalgamation is inadvisable on the material now treated of.

"The dead (oxidizing) roasting of sulphides as a preparation for pan-amalgamation improves the process by preventing the loss of quicksilver consequent on working raw sulphides, but it is not so successful as is generally supposed, or as theory would promise. The operation of roasting (whilst it apparently frees the gold from its combinations, increases, perhaps, the size of the gold particles, and eliminates the objectionable sulphur compounds,) seems to have a bad effect on a large part of the gold, putting it superficially at least in a condition unfavourable for securing contact with quicksilver. The extent to which the gold is thus affected varies curiously in different ores of apparently similar composition, and the conditions of roasting have also an influence on the result.

"For the amalgamation itself of the roasted ores various methods are recommended. One process used in the colonies is by working with large excess of quicksilver, and little water, and apart from contact with iron. The effect of grinding in iron pans seems in some cases much less beneficial than when amalgamation is conducted in stone arrastras. The use of gold amalgam in place of quicksilver, and the avoiding of contact with iron surfaces, was found most beneficial in experiments conducted by Mr. Stetefeldt, in Mexico, and a high percentage was extracted from a low grade ore.

"On the other hand, experiments on many gold ores in the New York ore-testing works gave unsatisfactory results by all these processes after roasting in a small reverberatory furnace, and nothing but chlorination was found effective when the material was rich in gold."

On most points the writer is perfectly in accord with the views that have just been stated in regard to raw amalgamation, and entirely so in regard to the inexpediency of roasting first, if only because of the extra cost added to the process when the ore is roasted, which scarcely any ore can bear, but he believes that, in certain exceptional instances, as regards the ore and as affected by locality, the process of raw-grinding is justifiable to a certain extent.

The writer thinks, as Dr. Foster pointed out in the discussion on Mr. Curtis' paper, that it is perfectly possible to extract a very high

percentage of gold from heavy pyritic ores and concentrates by this method, of which there are examples in the Pestarena ore in Italy (amalgamating it in Frankfort mills) and in some of the Transylvanian gold mines, where Tyrolean mills are employed for a similar purpose.

More frequently, however, one finds where an ore carries heavy pyrites it is not what one would term free-milling, and only a certain proportion, roughly speaking 50 to 65 per cent., can be got out by raw grinding, the process being generally accompanied by heavy loss of mercury and high working costs.

It is still more often the case, that mere grinding without previous natural decomposition (which can sometimes be assisted by adding a little salt to the ore pile and allowing it to weather for two or three months previous to treatment) will only extract a very insignificant percentage indeed, and roasting with salt generally will merely tend to make matters worse, both commercially and metallurgically.

The method of grinding, just alluded to, which is followed at Pestarena, in Italy, is in several respects unique, and cannot be passed over without special comment.

The ore, which consists of quartz, mixed with micaceous, graphite-schists, carrying iron pyrites, associated with a little arsenical pyrites, is first screened, and the lump ore, after being picked over, and if necessary cobbed, is crushed in a rock-breaker, and then delivered to Cornish rolls, whilst the screenings (mine fines) are trammed direct to the rolls, unless of too poor quality, in which case they are concentrated first by jigging.

The medium-size ore (resulting from cobbing) is washed and carefully picked over on tables, provided with settling-tanks to catch any fine pyrites that might otherwise escape, before it is sent to the stone-breaker.

The product of the rolls derived from these different sources is delivered to bins, connected with what are known as Frankfort mills, an improved form of arrastra.* 5 to 10 lbs. of lime per ton of ore is added to the charge to prevent sickening of the mercury. The action of sickening may be caused by greasy substances such as graphite, which, under the name of kish, is often used to cover the surface of castings, oil, grease, etc. Metallic oxides and finely divided carbonate of lead likewise cause flowing, and so do some soluble chlorides, owing to the formation of calomel on the surface of the mercury. Ferric-chloride being formed when ores are amalgamated in an iron pan with salt and sulphate of copper, will account

* The writer intends to make the construction and operation of these mills (for a description of which he is indebted to the kindness of Messrs. John Taylor & Sons) the subject of a separate paper, he will therefore pass over their description here, though of very great interest, without further comment.

for this action in some cases; where it is due to metallic chlorides a little metallic zinc added to the quicksilver will preserve it; when caused by graphite, which has been used to coat a mercury trough for instance, the only remedy is to carefully varnish it with shellac.

The quantity of ore treated in each mill varies from 10 cwts. to 1 ton per 24 hours, depending on the proportion of pyrites that it carries.

The report of the Pestarena company, for the year 1888-89, showed that 4,474·811 tons of wet ore, representing 4,345·443 dry metric tons, or 4,276·785 dry English tons, were milled in this way, at Val Toppa, during the 12 months ending June, 1889, with 20·5 mills, working 309 days. The results showed an extraction in bullion of 8 dwts. 23¾ grains, per English ton of ore. The bullion being 797·1 fine in gold, and 197·8 fine in silver. The fine gold, determined in the crude ore by assay, amounted to 8 dwts. 20 grains, per English ton, of which 7 dwts. 4 grains, were extracted by the mills, equivalent to a duty of 81·1 per cent., with a loss of 234 grammes of mercury per metric ton of ore treated.

The report of the same company for the year ending 1889-90 showed that 5,916·771 metric tons of wet ore, representing 5,724·004 dry metric tons, or 5,633·560 dry English tons, were milled at Pestarena (a separate establishment) during the 12 months ending June, 1890, with 25·9 mills, working 344 days. The results showed an extraction in bullion of 18 dwts. 8 grains per English ton of ore, the bullion being 742·4 fine in gold, and 252·2 fine in silver. The fine gold determined in the crude ore by assay, amounted to 17 dwts. 8⅓ grains per English ton, of which 13 dwts. 15¼ grains was extracted by the mills, equivalent to a duty of 78·77 per cent., with a loss of 230 grammes of mercury per metric ton.

Looking at these results, the question arises as to how far the extraction in grinding is affected by the fineness of the metal present entering into the bullion, the degree of concentration, and the grade of the ore, and it seems to lend confirmation to a point observed in other cases, that a low-grade ore, carrying fine metal (gold or silver) with a large percentage of gangue or sand in the concentrates, yielded in proportion a higher return than when these conditions are found reversed.

It may also happen that the loss of mercury is affected by the duty of the mills. The agents, Messrs. John Taylor & Sons, in their report on the Pestarena mines, for the year 1888, stated that the duty or percentage extracted of the assay contents of gold in the ore had increased, being 0·80 as against 0·78 last year; and the consumption of mercury had increased from 242 grammes to the ton during last year to 310 grammes per ton.

If the loss of mercury were due, as might be imagined, to the extra baseness of the ore, one would expect the yield to fall off, whereas the contrary is the case. The writer is inclined, therefore, to think that it may be rather attributed to extra grinding of the ore, which would tend to increase both the yield of gold and loss of quicksilver. Professor Le Neve Foster has remarked that on the whole more gold is extracted in these works in winter, when the water, coming from the mountains, is comparatively cold and clear, than when it is turbid and warm in summer. In July, 1887, for instance, at Val Toppa, the duty of the mills was 70·6 per cent., and varied from September to December from 91·7 to 88·5 per cent. Though it does not, in this respect, appear to follow an invariable rule in different years, this may be accounted for by modifying causes in different seasons, character of the ore and its general manipulation, as to amount of grinding, etc., and perhaps, as Dr. Foster thinks, the turbidity of the water, as well as the more rapid oxidation of the pyrites in summer, may partly account for these results.

The conclusion he draws that low temperature is not incompatible with good amalgamation in this instance, is no doubt perfectly true as well, since a certain amount of heat is generated by the mechanical action of grinding;* but of course this statement is not intended to apply to plate-amalgamation.

In Dakota, for example, where the thermometer sinks, at times, to 40 or 50 degs. below zero, provision is always made for heating the supply served to the stamps by passing the waste-steam of the engine through a steam-coil in the supply-tanks, and there is generally a steam-drum as well, running in front of the apron-plates, as an extra precaution.

Either very cold or warm water† would certainly be pernicious in plate-amalgamation. Speaking of the writer's own experience of grinding raw sulphurets in pans, he made some careful experiments in 1884, treating a quartzose ore, carrying on an average 2½ per cent. of pyrites, which was concentrated on end-bump tables. He put a lot (about 14 tons) through iron wheeler-pans, subjecting them to continuous amalgamation for 8 hours on somewhat the same plan as the American Boss system (so called after its inventor Mr. H. P. Boss), and found that though the concentrates originally assayed only 1 oz. 16 dwts. 18 grains, the tailings still contained 17 dwts. 12 grains per ton.

* Dr. Foster roughly measured the increase in temperature in course of grinding in a Frankfort mill at Pestarena, and found that an average rise of about 3° Fahr. took place, but on the mill-bed itself the heat must have been much greater.

† The-pan amalgamation of silver ores, it is true, is facilitated by a high temperature (live steam being often introduced to warm the pulp), but the conditions in that case are entirely different.

In the Black Hills, it has long been the fashion to treat blanket concentrates in pans. Dr. Hofman states* that when panned down for experiment, these gave 20·5 per cent. cleaner concentrates, assaying 40·18 dollars per ton, but when amalgamated in a wheeler-pan, only yielded 56·9 per cent. of their gold contents.

In Australia, again, the witer has constantly found that battery-pulp concentrated on percussion-tables and buddles, up to 60 and 75 per cent. of pyrites, when ground in berdan-pans, only yielded 50 to 60 per cent. of its gold in the first grinding, though about half as much more might be extracted by a repetition of the process at much extra time and cost.

The Warden in his official report on the Charters Towers gold-field for 1878, page 14, remarks in connexion with this subject: "There can be no doubt that in many instances more gold escapes with the tailings than is in the first instance obtained by the crushing mills. For support of this, I am aware of several hundred tons of tailings having been put through the stamper boxes as a trial, and the result obtained was 15 dwts. to the ton. If such a yield is obtainable in such a manner, I may well ask the question, what must be the yield of tailings when properly treated with the most approved gold-saving appliances? Practical men of long experience inform me that tailings† on this field, under proper treatment, will yield on the average 9 ozs. 5 dwts. per ton." The writer does not think that the average loss of gold at the present time in the pan-tailings of Charters Towers is much less, however, than 14 dwts. to the ton.

The only remedy that has been applied to any large extent is grinding and re-grinding. The re-grinding of these tailings (after the miner has relinquished his interest in them), was at one time a regular recognized business, and is doubtless still carried on to some extent at some of the mills.

This, no doubt, accounts for no small portion of the gold, indirectly supposed, if not alleged to be stolen by the miners working in some of the mines, or by the mill hands, an explanation the writer does not for one moment believe, though it may certainly be a convenient red herring to draw across the scent to divert investigation from channels it might possibly follow with more advantage to the general mining community and the industry at large.

There may be small isolated thefts of specimens, but, taken as a body, the writer has always found the Queensland miners an uncommonly honest set of men, and it is quite impossible that 18,593 ozs. to 15,169

* "Gold-milling in the Black Hills," by H. O. Hofman, *Trans. Am. Inst. Min. Eng.*, vol. xvii., page 538.

† The allusion here is evidently to pan-sludge after concentration and grinding.

ozs. (the excess of gold purchased by the banks over the mill returns in 1889 and 1891*) could have been illicitly obtained and secretly crushed.

As the mills do not publish any record of the gold got from the source referred to (that the writer is aware of), there is no check on the matter, and the miner who has an interest in a mine or claim (and there are few who have not) may more probably be the real sufferer,† whilst being branded actually as the culprit.

In pan-amalgamation, the flowering of the mercury which Mr. Curtis has alluded to, even when heavy pyritic concentrates are being worked, can generally be avoided with care, by selecting a suitable class of pan, grinding the ore to sludge before adding the mercury, and raising the shoes (if wheelers are used) off the bottom, during amalgamation (so as to allow the muller merely to act as an incorporator), keeping the pulp of the proper consistency (neither too thick nor too thin) and using chemicals judiciously, adapted to the particular conditions that obtain.

If the pulp gets too thick, the inevitable loss of mercury in treating ore in berdans is likely to be greatly intensified; using wheelers, the charge should be about the consistency of cream, and to avoid loss, the settler must be properly looked after.

A consideration of the MacArthur-Forrest process, also forces one to consider the losses of the precious metals, (more particularly gold in solution), that may occur through the indiscriminate use of chemicals, such as cyanide. Other re agents such as salt, iron, sulphate of copper, sal-ammoniac, and caustic soda and potash, are used by different millmen, in certain cases, in the amalgamation of gold and silver ores.

Cyanide keeps the mercury quick and lively, and the last-named reagents neutralize greasy substances introduced into the pan, for which wood-ashes are also employed. Caustic lime is sometimes used with roasted ores which contain a great deal of cupric chloride, to reduce it to cuprous chloride. Sodium amalgam is useful, especially for ores containing binoxide of manganese, to prevent flowing, which that metal, as well as copper and lead, are more especially liable to occasion, by debasing the amalgam.

Blue-stone, salt, and iron assist the decomposition of certain silver ores, and an acid solution of the former salt, when there is a large proportion of clay in the ores, which causes it to ball-up and carry off globules of mercury, probably acts on the clay like alum, tending to promote its precipitation and prevent loss.‡

* *Charters Towers Gold Mines*, by L. W. Marsland, page 197.

† *Vide* remarks on page 96.

‡ Tenth Report, *Census of the United States*, 1880.

Dr. Raymond and others have frequently pointed out that some gold is lost in solution in ordinary plate-amalgamation, but if cyanide is the active agent in attacking gold and silver which it is represented to be, it stands to reason that very large hitherto unsuspected losses may occur in certain instances, especially when the gold is naturally in a fine condition.

We do not know how large these losses may have been in the past, because the writer ventures to say that there is no mill in the world where amalgamation-tailings are regularly tested for gold in solution, but if we have not examined the question hitherto, it is one that deserves to be looked into in the future where cyanide is used to any extent. A little red-oxide of mercury dissolved in the cyanide, will often greatly assist amalgamation, and for the recovery of the dissolved gold, a little zinc amalgam may be added to the pan towards the end of the operation.

Raw-grinding is still the favourite process in North Queensland, and it is likely to remain in fashion there for many years, for good reasons, or otherwise, although it has been proved to be a more or less mistaken idea.

The very fact that it pays local chlorination-works to purchase and treat the pan-tailings after the first grinding, shows that it cannot be otherwise than sheer waste of money with some of the Charters Towers ores to incur the first expense of grinding at all. Amongst the reasons alluded to may be mentioned:—

1. The heavy loss of capital involved in making a radical change in a field where all the mills are laid out specially for grinding, as the large number of pans which are now running would only be worth the value of old iron if they were thrown out; affording a good illustration of the importance of choosing the proper kind of plant in opening up a new district, making due provision for probable changes in the class of ore likely to be met with in depth.
2. The fact that some of the Queensland ores contain considerable amounts of base metal, especially lead, and these can be treated at some profit by grinding-milling, whilst it has yet to be determined how far this class of stone can be otherwise worked to better advantage; smelting being out of the question at present.
3. The difficulty of overcoming the general prejudice on a gold-field against any new form of procedure.
4. The fact that as many claims are worked by prospectors, and as many of the mills are run on custom-work, chlorination-works would have either to work a number of small lots separately, which would be costly and impracticable, or else buy the ore outright, which introduces the next difficulty.

5. That the miner, with a wisdom born perhaps of experience, likes to follow his gold and look after it himself from start to finish, and would not be convinced that selling it on assay the advantage did not gravitate to the side of the mill. It is only when he has followed it just as far as he can, and scraped every crevice of the mill himself (to see that there is not one atom of amalgam lost), that he is ready to let the tailings go out of his own control and sight.
6. A change would perhaps temporarily throw a small number of men, accustomed to grinding-milling out of employment, and introduce the necessity of specially skilled superintendence, so that any alteration in the existing *régime* might affect local capital and local management in these respects, and as the gold-fields policy of Queensland, when controlled by local influences, has up to the present turned apparently on the exclusion of everything British except money, the change is not likely to come yet awhile.

The chief sufferers, of course, are the miners themselves (who are mostly shareholders in the mines but not in the mills,) and outside capital introduced into the district, representing at present, the nominal amount of £1,737,268 out of a total of £3,774,400, invested in mines, in the colony of Queensland. Both the miner and capitalist are, it would therefore seem, paying to maintain a wasteful system, which benefits comparatively few people, by which the country loses a large amount of gold, which would be better in circulation in the pockets of the community, than in the river bottoms.

In districts, like Charters Towers, no doubt, we shall ultimately see most of the mills adopting the principle of simply catching all the free-gold they can in the battery and on plates,* concentrating thoroughly, and disposing of the pyrites to central chlorination-works, merely utilizing the pans as they stand for treating the tailings of the concentrators after they have been classified, whilst large outside mines, which have ample stocks of pyrites, will, in time, put up chlorination-works of their own with advantage; but smaller concerns will no doubt do best to stick to grinding and suffer the loss for the time being.

Where gold ores must be ground, it would be better, in the writer's opinion, to grind in Frankfort mills, or to substitute Chilian mills for

* A few pans might, of course, be kept for occasional use, to deal with ores that could be advantageously treated by grinding, under conditions described.

berdans, and amalgamate on copper plates outside, grade the pulp in classifiers, concentrate on efficient tables, and treat the concentrates by chlorination, or else where pans are employed for the treatment of sulphides (concentrates), to grind continuously in large-sized wheelers, with wooden sides (in place of the small iron ones that are in general use), and amalgamate afterwards with the mullers raised; or as another alternative, distribute the ground-up pan-sludge to berdans, discharging into large settlers provided with proper drags.

The writer's reason for this opinion is that even an ordinary wheeler* is a far faster grinder than a berdan, whilst the latter machine seems to be a better gold-amalgamator.

The writer ventures, however, to think that the statement that grinding in the colonies is the outcome of poor concentration, ought to be reversed, and for this reason, that a certain proportion of sand must be present to brighten the liberated gold and lighten the charge.

Another special point to observe is that a berdan grinds and amalgamates better, and naturally wears out less, if the drags are made of softer iron than the pan itself.

Hardwood drags may be used if the pan is intended simply as a machine to amalgamate and collect amalgam in. It is of course a first essential that the pan should be speeded right, and inclined at the proper angle from the vertical, depending on its diameter. The gold is amalgamated by the pan being allowed to run full for some time without overflow, water being afterwards turned on from a pipe to expel the ground-up sludge.

Eight berdans are about an ordinary allowance to a battery of five stamps (which crush on an average 2 tons per head per day) for dealing with the concentrates from the ordinary run of Queensland ore, carrying 2 to 6 per cent. of pyrites, though this allowance is often exceeded for re-grinding and treating extra heavy ores.

The cost of 32 berdans (the number we may assume on this basis that would be required) for a 20 stamp battery, including the shed to cover them, erected in running order at Charters Towers, would not be far short of £1,200 to £1,300, and the 32 pans would have a working capacity of something like 28 to 30 tons of sand and concentrates weekly, allowing, of course, for variations in the method of treatment, degree of concentration, and amount of re-grinding to be done.

* The writer alludes to the small cast-iron type in common use, not the wooden-sided combination pan used in silver-milling, between which and a berdan, in the matter of grinding, there can be no need for comparison.

The cost of this treatment may be estimated as follows :—

	Per Berdan. £	s.	d.
Wages per month	0	18	2
Wood „	0	11	10
Mercury „	1	2	9
Castings, wear of belts, etc., per month...	0	9	1
Handling sludge	0	3	3
Oil and chemicals	0	1	9
Total cost	£3	6	10

or equal to 2s. 6d. per day.

Assuming that each pan grinds about 3 cwts. per day (dry weight) of 60 per cent. concentrates, it is equivalent to saying that one pan grinds 201⅜ lbs. of clean concentrates per day, or 1 ton of pure pyrites in, say, 11 days, at an actual mill cost of £1 7s. 6d.* for a yield of 50 to 65 per cent. of the gold which the concentrates contain.

Taking the cost of the grinding plant (covered in) at £1,200, and reckoning interest on the spot at 4 per cent., we must add to the above costs £48 per annum, or 7½d. to 8½d. per ton of concentrates ground, equivalent to about 11½d. per ton of pyrites, making a total grinding cost of £1 8s. 5½d.† per ton of clean pyrites.

Looked at in another light, the actual working cost of grinding the Charters Towers concentrates in berdans may be reckoned at 2s. per crude ton, added to the ordinary milling-costs, in a 20 stamp battery with a duty as stated.

The Northern Miner of February 14th, 1889, gives an interesting account of a trial made at Charters Towers by Mr. Millet, a well-known miner, at the Mary Louisa mill, on an unusually heavy pyritic ore, from the 1 and 2 West Wellington mine. Eight tons were crushed as an experiment and yielded 4 tons 4½ cwts. of concentrates, 2 tons 4½ cwts. of which were taken for trial. This was distributed between the wheelers and berdans in the proportion of 1 ton 2 cwts. treated in the former, and 1 ton 2½ cwts. in the latter. The wheelers gave 2 ozs. 3 dwts., the berdans 2 ozs. 6 dwts. of gold. The berdan put through one wet ton per week, at a mill charge of £2, while the wheeler put through 5 to 6 tons, at a charge of £9, or about the rate of £1 12s. 9d. per ton, leaving a margin on charges in favour of the wheelers of 7s. 3d. per ton. The total extraction by milling and grinding‡ was 31 ozs. 14 dwts. from the 8 tons of stone treated, not counting the gold left in the pan-sludge, or the

* These figures are, of course, liable to fluctuate somewhat (up as well as down), depending on the relative percentage of sand, sulphides in the concentrates, and the actual percentage of the latter in the crude ore. † *Ibid.*

‡ Even with docile ores this seldom reaches more than 76 per cent. of the total gold in the stone at Charters Towers.

2 tons of concentrates left unground. If these latter had been ground the yield would doubtless have been increased by another 4 ozs., or an equivalent of about 4½ ozs. per ton.

As over 55 per cent of the total gold was got by amalgamation in the battery it would be extremely interesting to know what the pan sludge contained after grinding.

The extraction, it will be noticed, per ton of concentrates, was 2 ozs. 0 dwt. 20 grains with the berdan, as compared with 1 oz. 19 dwts. 4 grains with the wheeler, or a difference of 1⅝ dwts. in favour of the berdan, and valuing this gold at 3s. 5d. per dwt. it really represents a saving of 5s. 8d. per ton, off-setting the smaller mill charge employing the other pans.

Now, if we assume the above saving to be fairly representative (as regards the two methods of treatment), estimating the cost of operating the berdan at 3s. 5d. per ton in excess of the cost of the wheeler (an amount seldom exceeded in a 20 stamp mill), it will be apparent that the advantage to the mill owner, crushing stone of his own, under existing conditions, would rest on the side of using the berdan, as he would gain the difference between 3s. 5d. and 5s. 8d., netting a profit of 2s. 3d. per ton. As there are chlorination-works already established at Charters Towers we can further compare the cost of grinding with that of chlorination in this same district.

Mr. Brown, in a letter published in *The Northern Miner*, October 23rd, 1886, gives the cost of vat-chlorination as carried on by the North Queensland Pyrites Co., at the Burdekin works (of which he was formerly manager), as follows :—

Cost of roasting 15 tons of ore at Charters Towers in an ordinary reverberatory furnace :—

	£	s.	d.
Labour, 6 men	18	18	0
Firewood	7	16	0
Water	0	10	0
Salt	1	0	0
Kerosene	0	7	0
Total cost	£28	11	0

or £1 18s. 0d. per ton.

Cost of chemicals and labour chloridizing 26 tons of ore :—

	£	s.	d.
Management	6	0	0
Labour, 2 men	6	0	0
Water	1	0	0
Salt, 3 cwts. at 6s.	0	18	0
Manganese, 2¼ cwts. at 5s. 6d.	0	13	9
Sulphuric acid, 416 lbs. at 5s. 6d. ...	5	4	0
Sulphate of iron	1	10	0
Kerosene	0	5	0
Total cost	£21	10	9

or 16s. 7d. per ton ; making the total cost, £2 14s. 7d. per ton.

The capacity of the furnace represented about the weekly capacity of the works, viz., 15 tons, and in the period ending September 1st, 1886, they paid 2s. in dividends* upon 740 tons of ore treated; £450 additional was invested out of profit in new plant, whilst the total capital cost of the works, up to the date in question, appears to have amounted to only £2,830 5s. The returns from the ore (pan-sludge) treated was 1,466 ozs. 8 dwts., or over 2 ozs. per ton. Owing to the success this trial plant had achieved it was proposed in 1886 to quadruple its capacity at an outlay estimated at between £4,000 and £5,000 additional, and Mr. Brown, in the letter previously alluded to, estimated that with an improved roasting-furnace† the time was not far distant when he would be able to treat a ton of ore at a cost of £1, working on a larger scale.

Mr. Brown goes on to say: "This field is exceptionally favourable for the establishment of extensive chlorination-works owing to there being a large percentage of float-gold, which is unfavourable for amalgamation, but is easily and perfectly extracted with chlorine. The time is close at hand when chlorine will do away with such extreme amalgamation as is now in practice, as the gold will be obtained much cheaper with that agent. Concentration is a matter which is much overlooked at present, and there is a large percentage of fine mundic passing off with the quartz-sand that could be easily saved by improved machinery, but this will not receive much attention until there is a demand for the concentrates, when it will be looked to. The large heaps of quartz-tailings, now looked upon as worthless, will be reduced to one-half their present value, and will pay handsomely under the chlorine process."

Now, if we assume on a liberal estimate that 80 per cent. of the gold in the pyrites can be extracted by a double grinding, and on a low estimate 85 to 95 per cent. by chlorination, we find that:—

	Per Ton. £ s. d.		Per Ton. £ s. d.
The cost of chlorination per ton of clean concentrates (page 99) amounts to	2 14 7	to	1 0 0
Add to this interest on a capital outlay of £2,800 to £7,800 at 4 per cent. on a capacity of 740 and 2,960 tons treated per annum	0 3 0¼	„	0 2 1¼
The total cost of chlorination is from	£2 17 7¼	„	1 2 1¼

Now, the cost of grinding per ton of clean concentrates has been shown to amount to £1 8s. 5½d. Therefore doubling this sum for re-grinding,

* On a nominal capital of £10,000, representing a net profit of £1,000 in eleven months, the year commencing August 12th, 1885, and ending September 1st, 1886.

† Which would reduce the cost of roasting to 10s.

the cost of obtaining 80 per cent. of the gold by this process, under the most favourable circumstances, will not be less than £2 16s. 11d., as compared with a yield of, say, 90 per cent. (to put it low) at a cost of £2 17s. 7¼d., under the most adverse conditions of chlorination.

If, in fact, we assume clean concentrates to run only 2 ozs. to the ton, the difference merely in yield at the average price of Charters Towers gold, will represent a money value of 13s. 8d. saved by chlorination, or a net gain of (13s. 8d. — 8¼d. =) 12s. 11¾d. per ton. With higher grade concentrates, and the costs reduced even to £1 10s. (as they easily ought to be), the gain with chlorination would be simply enormous. The question whether it is expedient or not to incur large gross cost in treatment for the sake of close extraction depends, in fact, not only on the quantities to be dealt with, but likewise on the grade of the material treated, it is evident that the richer this is the more important it becomes to save close.

With a single grinding, the results are scarcely less disadvantageous to the pan process, as the mine owner loses the difference between 65 per cent. and 90 per cent. of the gold in the pyrites, or, say, 10 dwts. per ton (worth £1 14s. 2d.), dealing with 2 ozs. sulphide ore.

The Charters Towers pyrites works, situated at Charters Towers itself, employ the vat process in the same way as the Burdekin works. The only peculiarities with regard to them are that they manufacture their own sulphuric acid on the spot, and employ a unique form of roasting-furnace—an idea which originated with Mr. D. A. Brown (who has been already alluded to), their able and energetic manager.

Mr. Brown conceived the plan of building his furnace on the side of a hill, possessing approximately the natural slopes requisite for the purpose he had in view, viz., to economize labour in manipulation and save as much fuel as possible (by utilizing the combustion of the sulphur in the ore, for roasting) without sacrificing the sweetness of the roast.

The furnace is shaped like a Fortschaufelung, with a number of doors on each side throughout its length.

To break up any lumps in the material (pan-sludge) to be roasted as it comes to the charging-floor it is first passed through a Carr disintegrator, driven by a pulley, provided with a coned friction-clutch. Below the disintegrator a hopper discharges the ore into three vertical circular openings, at the head of the hearth, each fitted with an Archimedean screw, driven by bevel-gearing. The speed of these screws regulates the feed, and delivers the ore into channels, which run down the upper part of the hearth.

Speaking from memory (as the writer has mislaid some notes he made when he visited the works in 1889), the hearth is about 600 feet in length, and has three different grades, commencing with 35 degs. at the head of the furnace, for a few feet, continued on for some 20 feet at a grade of 27 degs., and for the remainder of its length sloping at 18 degs. The fireplace and wrinkle (with working-openings on each side), back-up, and form the foot of the furnace.

Just below the brickwork of the hearth, three parallel lines of 6 inches fireproof stoneware pipes are laid the greater part of its length, connected at the two ends by bends, so that the artificial draught produced by a Roots blower, at the head of the furnace, circulates backwards and forwards three times through the pipe till it is finally discharged in a superheated condition, into the fireplace. The downpipe of the blower is provided with a gate-valve to prevent the furnace gases being drawn back into it when the blower stops.

The roof of the furnace is very low and flat, but presents a succession of 6 feet to 8 feet span transverse arches, with their curtains set crossways to the length of the hearth (which is about as wide as that of an ordinary reverberatory furnace) to throw the flame down on to the ore.

It was a bold project, which deserved success, though at first sight it would appear to be beset with many practical difficulties, in controlling the ore during its descent, so as to obtain a dead sweet roast during its gradual passage, from the top to the bottom of the furnace; and at the same time to avoid large losses in dust, not to mention difficulty, from the superheated gases entering the lead-chambers.

Still, from the fact that the furnace has been in active operation almost continuously from the time it was properly started, these objections would seem to have been overcome. Having lost sight of the matter lately, through pressure of various business engagements, the writer is unable to say what measure of success this system has actually achieved.

In making a preliminary trial of the furnace, Mr. Brown stated that he was able to roast 50 tons a week, with a consumption of 1 cord of wood to 10 tons of ore roasted, and he expected to increase this by some alterations that were in contemplation to 70 tons weekly without any increase in fuel burnt, anticipating that 1 cord of wood would be sufficient to dead sweet roast 15 tons of ore.

(*To be continued.*)

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THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.

By A. G. CHARLETON.

Part III.—Silver.

Wet and Dry Pan-amalgamation and Lixiviation.

Pan-amalgamation, as applied to silver ores, is always preceded by stamping the ore in a battery. If conducted wet, the pulp is collected in tanks, from which it is shovelled into the pans (an arrangement which might, the writer thinks, be improved upon), or else, if arranged on the Boss system (much in fashion lately owing to the saving in labour and other advantages claimed for it) the pulp is run straight through a series of pans without any intermediate settling. If conducted dry, the ore is taken direct from the cooling-floor of the roasting furnace (which forms an essential part of the plant), and is charged into the pans afterwards.

Pan-amalgamation is most extensively used in the Western States of America for the extraction of silver from its ores, and with such exceptions as have been or will be alluded to, is, the writer thinks, likely to retain its position for some time to come.

It broadly divides itself, as will be presently seen, into:—

1. The Washoe process, in which the ores go direct from the tanks to the pans wet, the amalgamation being generally assisted by the use of chemicals, chiefly salt and bluestone.
3. The Reese River process, in which the ore must be first dried: (*a*) on a drying-floor heated by waste-steam or furnace gas; or (*b*) in revolving drying-cylinders or shelf-kilns, which are steadily coming more into fashion. Roasting (usually with salt, which is mixed in the battery, or between it and the furnace) follows drying, and the ore is ground in pans in the same way as in the wet process.
3. The continuous Boss process.

The Washoe Process.

In an ordinary wet-crushing silver mill, the ore is brought in cars to the top of the mill-building, where it is dumped over the top of the inclined grizzley or screen on to the crusher-floor. All the small pieces pass through the grizzley into the ore-bins below. The coarse rock is shovelled into the jaws of the rock-breaker, which are on a level with the crusher-floor. The ore crushed to walnut size in passing through the rock-breakers falls into the ore-bins, and thence goes to the automatic feeders (behind the stamps), passing through inclined shoots controlled by gates. The automatic feeders being kept full, ensure a uniform supply of stone being fed to the stamps as fast as needed.

The finely stamped ore, known technically as pulp, suspended in water, flows into large settling-tanks, where the excess of water is drawn off, while the thick pulp remaining is shovelled in regular charges into a row of amalgamating-pans, in which it is ground for several hours, first with salt, bluestone, and other chemicals, and afterwards amalgamated with mercury, with the mullers raised. The contents of the pans are run into large settlers (when the previous operation is finished) placed below, and in front of the pans, in which the pulp is thinned by additions of water and gentle agitation, and all the quicksilver with the precious metals in the form of amalgam, settles to the bottom. The pulp is gradually drawn off from the settlers (through holes fitted with plugs at different levels in the side) and flows to waste. The amalgam is strained from the excess of quicksilver, retorted to drive off what remains, and the resulting gold and silver cake is melted into bars. The gold and silver contained in the sulphides, which will not yield to the above treatment, is sometimes caught by concentrators (Frue or Embrey) which receive the waste pulp-tailings from the settlers. A clean-up pan generally forms part of the plant.

The old method usually employed in silver-milling was to crush coarse in the battery and grind fine in the pan, but as this involves greater power, greater wear of castings, greater loss of mercury, and not always better results, the system of crushing fine in the battery (keeping the shoes barely off the dies) has lately come into practice. By this means more gold and silver will sometimes be extracted, since ore will naturally break where there is most mineral, and the fine comminution in the battery will generally disengage most of it, the consequence being, that as quicksilver has a preference for gold and silver, it will amalgamate with them, rather than take up base metals (which render it inactive), which it is forced to do by excessive grinding.

The modern amalgamating-pan is a growth from the old arrastra, and though its construction is quite simple it presents a variety of forms. The pan holds from 1 and 1½ to 2 tons of pulp, and generally revolves at about 60 revolutions per minute; the gearing underneath is open and plain, the muller is raised by a left-hand screw on top, the hand-wheels of which should be large (the jam-wheel being no smaller than the screw-wheel), as it frequently requires a greater application of muscle than the latter, and when the machinery is in fast motion it is inconvenient to adjust a small wheel under a large one. The most important feature of the pan is the pulp-current, which often receives but little attention, and though simple in principle is not always understood, and its neglect may cause serious loss.

These currents must be uniform and regular to ensure uniform work, and strong enough at the bottom of the pan to carry the quicksilver. The motion of the muller makes a current by throwing the pulp to the outside as it advances, which then rolls up at the side and falls over towards the centre, and down through the central opening, in and under the muller, to be thrown outwards again from the bottom. This so far cannot be improved upon, but the wings are needed, to give the pan capacity (by preventing the pulp from running too high up at the side), to accomplish which, they should have the shape of an inverted ploughshare.

Having naturally a good current above the muller, we have only to work in unison with that underneath it, which will depend on the design of the muller and setting of the dies.

The pans vary in diameter from 4 feet to 5 feet 6 inches, and have generally a cast-iron flat bottom with wooden sides. They ordinarily hold 1,200 to 1,300 lbs. of ore, and three is the usual number allowed per battery of five stamps, but sometimes two will be found sufficient. Each pair of pans requires one settler. In some districts copper plates are introduced into the pan, and much of the amalgam is found attaching to these, but the more usual system is to employ settlers entirely for the collection of the quicksilver and amalgam, after the pans are discharged. While the pulp is being worked in the pans, which usually takes 6 to 8 hours, steam is introduced to heat the mass and promote the chemical reactions; sometimes live steam is introduced direct from the boilers into the charge, but more commonly the pan is furnished with a false steam-bottom, and heated with the exhaust from the engine. The bottom of the pan is protected by cast-iron dies, and the muller is furnished with adjustable shoes, so that the wearing surfaces are renewable if it be necessary to grind the ore. The shoes and dies can be brought together when grinding by means of the hand-wheel, and

screws on top of the spindle, or the muller can be raised above the dies for circulation and mixing only.

The screw on the settler-driver, should, unlike the pans, be right-handed, for besides being more convenient in case of a belt slipping, the power applied to turn the screw, helps the muller to revolve. A settler should never be allowed to foul by an accumulation of heavy matter at the bottom, it is a positive preventive of good work. It is, however, easier to advocate this than to do it. An apparently natural remedy, viz., a liberal use of water, tends rather to aggravate the difficulty; there is a point in the thinning when the quicksilver will be precipitated, but the heavy sand be held in suspension.

If, after the charge is run out (which should leave about 8 inches of pulp in the settler), a pan is drawn and no water is added for half an hour, the warm charge will gather and carry the heavy sand; now enough water only is added to reduce it to the appearance of still some thickness, and this is all the water that is used in the charge. A horn spoon will show its success in advance of results.

Settlers are generally made with wooden sides 8 feet in diameter inside the staves, an automatic syphon-tap being provided for the discharge of the quicksilver and amalgam. Around the bottom a groove is cut, starting from nothing on one side, and gradually deepening to the syphon-tap opposite, in which all the quicksilver is carried to the outlet.

The muller-plate attached to the driver-arms, is shod with wooden plough-shoes, which are sometimes, however, attached direct to the arms themselves. The speed of the settler is generally about 15 revolutions per minute. Concentrators for a silver-mill must of necessity be simple and capacious. Good agitators (shovelled out often) are profitable. In some cases sand sluices are very effective, consisting of a broad sluice 20 to 24 inches wide, in which at intervals of 8 or 10 feet vertical strips are fixed at the side to hold movable riffles. The riffles (battens of wood) are laid in, and the sands run over them for a time (say one or two hours), when another course of riffles $\frac{1}{2}$ inch thick or less is laid on the first ones.

This is repeated until the sluice is full, when it is shovelled out, meantime allowing the sands to run through a duplicate sluice, at the side. Such sluices should have a grade of about $3\frac{1}{2}$ inches per rod to keep them under control, as by starting with a thin riffle at the bottom a strong current may be produced, whereas the introduction of a thick riffle will give a deadened current.

These sluices are an advantage where blankets are profitable, and if followed by blankets relieve the latter from much coarse, heavy material.

A blanket sluice should have a grade of about 3 inches in 7 feet. The stock of mercury in a silver-mill should be large to begin with. In a dry-crushing silver-mill the loss is usually from $\frac{1}{2}$ to $\frac{3}{4}$ lb. per ton of ore. A 10 stamp mill usually requires from 200 to 250 lbs. of mercury per month to make up the loss. The quantity needed in stock depends on the richness of the ore, but is approximately 1,500 lbs. in the pans, 1,500 lbs. in the settlers and in circulation, and 1,500 lbs. locked up in amalgam, so that a total stock of 2 to 3 tons would be necessary for starting with.

Roasting Milling.—The Reese River Process.

After passing the rock-breaker, the ore is dried by passing through a continuous revolving-drier or shelf dry-kiln beneath the breaker, the dried ore being taken by car or else run through shoots (lined with sheet iron and regulated by gates) to the automatic feeders, if the fall admits of it. The stamps are fed while the ore is still hot, the pulverized product being conveyed to the elevator, by which it is carried to the iron storage-hopper of the roasting-furnace. In the furnace the ore, with the addition of common salt, is desulphurized and chloridized, thus preparing it for the pans and settlers. After roasting, the ore is spread on a cooling-floor, and is taken in cars as required to the pans. Amalgamation follows on the same plan as in wet crushing-mills. In old type mills it was formerly the practice to employ drying-floors of boiler or cast-iron, from which the ore was shovelled to the stamps, in place of the more modern arrangement of an automatic revolving-drier.

The Boss Process.

This process marks a new epoch in the milling of silver ores in the United States and in Mexico, as it presents claims to superiority in many respects over the old system of pan-amalgamation. The large saving in labour and fuel, increased cleanliness, reduced wear and tear, and other features that will be mentioned later on, combine to make it a favourite with mill-men. It does away with the large pulp settling-tanks and consequent shovelling and handling of the pulp, which is a serious item of cost in ordinary wet treatment; it saves the erection of the tanks and the space they occupy; and no slum-pump or agitators are required.

The buildings and cost of erection for a continuous mill are less expensive than for an ordinary one, as they require less grading and retaining-walls and cover less area. The ore passes through the grizzley and crusher in the usual manner and down the automatic feeders to the stamps. The pulp flows from the battery through pipes to the special

grinding-pans (the product of ten stamps passing through two in succession). The pulp is then conveyed by pipes to the first amalgamating-pan, and flows continuously through the lines of pans and settlers. The tailings are run off and led over concentrators. A special feature of this process is that the pulp in the amalgamating-pans is always kept thin, instead of being about the consistency of thick cream, as usual in the ordinary pan process. The quicksilver is charged to the pans by means of pipes from the distributing-tank, and the amalgam flows direct to the strainer. The chemicals are supplied to the pans by two chemical-feeders. Steam syphons are provided for cleaning out the pans, and for conveying the pulp past any pan when it is necessary to cut it out of the series for repairs. The main-line shafting runs directly under the pans and settlers, each of which is driven from it by a friction-clutch. This arrangement of separate clutches for each pan and settler is very convenient, as any number or any one pan and settler can be stopped in case of accident for cleaning out, without having to stop the whole line.

All the water from the batteries must pass through the pans, so that all the slimes are treated; there is less loss of mercury and a true sample of the tailings can be obtained, a matter of much greater difficulty with the old method. Heating by exhaust steam is stated to be one of its economical features, obviating any strain upon the pan, which is heated indirectly through the hollow steam-bottom. Where changes were made from live to exhaust steam in some mills it is said to have saved as much as £2 10s. 0d. to £3 2s. 6d. per day for cord wood.* By using special grinding-pans, the ore can be crushed through a coarser screen in the battery, and the finer grinding can be afterwards accomplished in the pans, thus obtaining increased capacity.

Though it does not pretend to cope with rebellious ores which are unsuited to such treatment, the Boss process is without doubt a great improvement over ordinary pan-amalgamation in tanks. If the latter process be employed, the tanks are filled in succession; the pulp being conveyed to them through a launder, by means of which the supply is cut off as each vat becomes full. Arrangements should be made to settle as much of the mineral as possible by allowing the water to circulate through the empty tanks before passing to the slum-pit outside, the escape or tailings water being turned into each tank after emptying it of sand. Each tank in turn thus receives the water after passing through the other tanks, and becomes the final one of the series. In some mills a certain number of tanks are kept employed in settling the sands, while the

* Messrs. Fraser-Chalmers, *Catalogue, No. 4*, page 82.

remainder are used up for the slimes; in others the capacity of the vats is large enough to settle the sands and slimes together. Before charging the pulp into the pans it is usually shovelled into heaps on the platform in front of the pans, which is slightly inclined towards the tanks to drain the water back into them. In the pan-treatment a proper consistency of the pulp, a proper degree of heat, and clean quicksilver, are matters of the chiefest importance.

While charging the pan the muller for grinding is kept revolving and lowered, water having been previously run in so as to fill the pan to within 12 to 18 inches of the edge, and heated with steam. Some mill-men favour direct heating with live steam, others by means of a jacket or false bottom. The charge must be heated nearly to boiling-point by turning on steam again during the grinding. At the commencement of the grinding the pulp is thin, but after a couple of hours it will acquire the proper consistency for receiving the quicksilver, which becomes diffused (by the heat and grinding) in small globules through the mass. The pulp should be thick enough to cling to a wooden paddle dipped in to test it, showing particles of mercury evenly disseminated through it, so that the charge will carry the quicksilver in suspension. The salt (say about 10 lbs.) is added as soon as the pan is charged, and 2 lbs. of sulphate of copper (or whatever proportion is used), half an hour later. After the pulp is heated to about 180 degs. Fahr. steam is cut off, and the muller should be lowered gradually during the progress of grinding. When finished (after about 2 hours), some 200 lbs. of mercury are added to a 1,200 lbs. charge of pulp. The grinding is then sometimes continued for another half-hour or an hour, when the muller is raised and the pan run with the muller up, for 3 hours more. There is, however, less chance of flowering if the bulk of the mercury is not added until the grinding is entirely finished. A quarter of an hour before drawing the charge, sufficient water is added to fill up the pan, thinning the pulp thoroughly, so that it will flow readily out of the pan, and cooling it.

The work should be so arranged as to charge and discharge a pan every 6 hours, which gives it a capacity of about 2½ tons. The pans should be discharged in succession, not all simultaneously; that is to say, as soon as one or two pans have been discharged and refilled, after a certain interval (depending on the number of pans in the mill) the next pan or two should have completed their 6 hours' work, and be ready to undergo the same process.

When discharged, a stream of water should be directed into the pans,

to rinse them out thoroughly. Everything thus flows to the settlers, and through the partial dilution of the pulp, the quicksilver settles to the bottom, and is collected in the syphon. During the discharge of the pans the settler arms are kept revolving, and after a short interval a spray of water is turned on, and allowed to run till the settler is full. It is then turned off and the muller is allowed to revolve for an hour. This allows the quicksilver to collect and settle. An abundant stream of cold water is then let in and the settler is allowed to discharge through the different plug-holes, commencing with the top one, the operation being timed so that the bottom hole is reached just in time to receive the next charge. Once a week or oftener the settlers should be cleaned out, and the coarse sand and sulphides accumulating in the bottom are re-worked in the pans. Generally two settlers discharge into one agitator, and a constant stream of water should run into them. They collect some coarse sand containing a little quicksilver, amalgam, sulphides, and a quantity of iron which is worked-up in the clean-up pan. The floors should be kept as clean and free from dirt as possible. All drains should lead into the agitators, and unless the weather is too cold the quicksilver floor should be sluiced down with a hose daily.

Retorting Amalgam.

The retorts for retorting silver bullion are generally cylindrical or square with the corners rounded off, and containing shelves for several iron dishes. They should be heated to a bright cherry-red heat before commencing the retorting, otherwise it is difficult to drive off the last traces of quicksilver. A serious loss is entailed by a retort bursting, not an uncommon occurrence even with the greatest care. They must therefore not be fired too strongly, and must be strongly made and well braced. The mercury fumes are condensed by condensers acting on the Liebig principle, the quicksilver being caught in a bucket of water, into which the end of the pipe from the retort dips; care being taken that the water is not able to run up into the retort as it cools by the end of the pipe being too deep under water. After the retorting is finished it is advisable to leave the retorts to stand for several hours before withdrawing the bullion.

For cleaning quicksilver from impurities, which become mechanically mixed with it, the quicksilver strainer invented by Mr. H. H. Oakes is recommended by Mr. Eissler and is described by him in detail.*

* *Metallurgy of Silver*, page 159.

General Remarks, Ores, etc.

The ores of silver which can be successfully treated by the Washoe process are those in which the metal occurs in a condition which will be acted on by quicksilver, assisted by heat, agitation, and certain chemicals (chiefly salt and sulphate of copper), which produce a certain quantity of sub-chloride of copper, through the secondary action of the metallic iron present. The chloride and sub-chloride of copper (both of which are liable to be formed), tend to reduce any sulphides of silver present, by exercising a chloridizing influence upon them, and at the same time decompose the sulphides of lead and zinc. The sulphate of copper, moreover, enhances the amalgamating energy of the mercury by tending to expel the lead, and by causing the formation of a small quantity of copper amalgam.

A list of the chief ores and minerals containing silver would comprise the following :—

Name.	Composition.	Per Cent. of Silver when Pure.
Naumannite	Ag_2Se	73·2
Eukairite	$Cu_2Se + Ag_2Se$	43·1
Hessite	Ag_2Te	62·8
Petzite	$(Au\ Ag)_2\ Te$	41·8
Sylvanite	$(Au\ Ag)\ Te_2$	10 to 15
Argentite (silver-glance)	Ag_2S	87·1
Stromeyerite	$Ag_2S + Cu_2S$	53·1
Sternbergite	$AgFe_2S_2$	34·2
Miargyrite	$Ag_2S + Sb_2S_3$	36·7
Pyrargyrite	$3Ag_2S + Sb_2S_3$...	59·8
Proustite	$3Ag_2S + As_2S_3$	65·4
Stephanite	$5Ag_2S + Sb_2S_3$	68·5
Brogniardite	$PbS + Ag_2S + Sb_2S_3$	26·1
Polybasite	$9\ (Ag_2Cu)\ S + (Sb\ As)_2S_3$	68·0
Tetrahedrite (Fahlerz)...	$(CuAg)_2S + (SbAsBi)_2S_3 + (Fe\ Zn\ Hg)S$	variable.
Xanthoconite	$(3Ag_2S, As_2S_3) + 2\ (3Ag_2S, As_2S_3)$ ...	64·00
Fire blende	$AgSbS$	62·3
Freieslebenite	$Pb_2Ag_3Sb_3S_8$	23·8
Cerargyrite (horn-silver)	$AgCl$	75·33
Bromyrite	$AgBr$	57·40
Embolite	$Ag\ (Cl\ Br)$	61 to 71
Iodyrite	AgI	46
Native silver	... ...	100·00
Arquerite (native amalgam)	$AgHg$	34·8
Electrum (native alloy of gold and silver) ...	... ...	27 to 32·7

Minerals, etc., often containing silver in small quantity :—

Galena	PbS
Blende	ZnS
Pyrite	FeS_2
Chalcopyrite	$CuFeS_2$
Erubescite	Cu_3FeS_3
Mispickel	$FeS_2 + FeAs_2$
Altaite	PbTe
Clausthalite	PbSe
Nagyagite	(Pb AuAg) $(Te, S)_2$
Chivialite	$(Cu_2Pb)\ S + \frac{1}{2}Bi_2S_3$
Dufrenoysite	$PbS + As_2S_3$
Enargite	$3Cu_2S + As_2S_5$
Cupel bottoms, dross litharge sweepings, etc.	
Slags, etc.	

The presence of sulphides of iron, copper, lead, zinc, and antimony, interferes with the success of the amalgamation process in several ways. They foul the amalgam and check the reactions of the chemicals on the free-milling minerals, and carry off in their refractory combinations a portion of the silver which the latter contain. It often happens that while the upper decomposed surface-ores of a vein are free-milling, as depth is attained (beyond the decomposing action of the air and surface waters) they change in character, through sulphides and base metals making their appearance in the ore.

Occasional natural deposits of chloride of silver and some rare instances of native silver, unaccompanied by sulphides, form, with certain decomposed ores, the chief types adapted to free-milling; though the process being cheaper than roasting-milling, ores are sometimes worked by it, which should properly be roasted, but the percentage saved in such cases is correspondingly low. Ores carrying quite a large percentage of base minerals may be worked by roasting-milling, but in many cases it is more profitable (when the conditions admit of it) to treat such ores by concentration and smelting. The presence of certain minerals in combination may render the chloridizing roasting of an ore extremely difficult. The Silver King mine in Arizona may be cited as an instance of this, as mentioned by Mr. Aaron.*

The ore in question consisted largely at one time of fahlerz, chlorides, bromides, and oxides, in a gangue of quartz and heavy spar, and being of high grade it proved well adapted to treatment by the Kiss lixiviation process, for which it had to undergo a preliminary chloridizing roasting.

It was found that this ore sustained a serious loss of silver by volati-

* Report of the Director of the United States Mint.

lization during roasting ; an extra draught produced by opening both ends of the fireplace somewhat mitigated the difficulty, but it was only finally overcome by introducing steam into the furnace (as originally suggested by Dr. Percy) which successfully met the difficulty ; the volatile metal chlorides (to which the volatilization of the silver is mainly owing) being decomposed and converted into oxides, with the instantaneous production of hydrochloric acid. Unfortunately, however, as depth was reached in the mine the character of the ore changed. The proportion of chloride and tetrahedrite fell off, and zinc blende and galena became more abundant, and this led to far more serious difficulties.

The roasting became slow and tedious; while previously a charge of 5 tons could be well roasted in 14 to 16 hours, converting about 95 per cent. of the contained silver into chloride. The percentage of soluble silver in the roasted ore decreased also somewhat, causing richer tailings, and as the grade of the ore likewise fell off, a serious diminution in the output of bullion ensued. The ore, moreover, developed a tendency to ball and form crusts on the furnace walls. The balls were peculiar, being perfectly spherical and of all sizes from a pin's head to a marble, extremely hard, and consisting of concentric layers. The ore being crushed wet and received into settling-pits, no doubt operated disadvantageously in this instance.

When the ore was by no means at its worst, analysis showed it to contain 12 per cent. of zinc—equivalent to about 18 per cent. of blende, 6 per cent. of lead as galena, a good deal of antimony, a little arsenic, a very little iron and copper, and trifling quantities of cadmium, selenium, tellurium, and bismuth. The conjunction of antimonial and plumbiferous minerals with zinc blende tends in fact to make roasting difficult. The character of the gangue also exercises a great influence on the roasting. The presence of quartz is advantageous, spar or gypsum is not troublesome, but earthy carbonates are detrimental, and magnesia bad.

At one time the ore contained less quartz and spar than formerly, and more of the so-called porphyry of the district, which contains magnesia in abundance.

In the case of ore which balls in the furnace when roasted with salt, the usual practice is to roast without salt, to a certain stage when the salt is added and the heat increased; but the presence of metallic silver and the absence of a fair proportion of iron rendered this method inapplicable.

The next idea tried, that of roasting to complete oxidation without salt, and then chloridizing by an addition of calcined copperas and salt, has been used with a slight modification on some of the worst ores with good results.

Another successful plan was to mix a certain proportion of sand, about 7 per cent., with the charge. The sand used contained a little silver, being the coarser portion of a pile of rather rich tailings from previous concentration. The addition of one-third of clean quartzose silver-ore was found to act favourably, 95 per cent. of the chloride being got out in 24 hours with 3 ton charges.

Mr. Stetefeldt has lately introduced the plan of drying and roasting ore with gas, at the Holden mill, Aspen, Colorado, and at the Marcac mill, Park City, Utah, where lixiviation is employed for the treatment of the ore. The former plant was put in operation in November, 1891, and consists of four double shelf-driers, with one 6 feet diameter Taylor revolving-bottom producer, and one large Stetefeldt furnace, with a Taylor producer, also 7 feet in diameter. Mr. Morse, the general manager of the Holden works, states that, on a recent run of 4,631 tons of ore, 96·4 lbs. of coal were used per ton of ore roasted, costing 14·45 cents. The coal, consisting of a mixture of about equal proportions of Colorado Newcastle and Sunshine coal, costing 3·00 dols. per ton delivered at the mill. The composition of these coals are :—

		Fixed Carbon. Per Cent.	Volatile Matter. Per Cent.	Ash. Per Cent.	Water. Per Cent.
Colorado Newcastle	I.	55·9	35·9	5·4	—
„ „	II.	48·6	37·95	11·6	1·7
„ Sunshine	III.	48·0	43·0	7·5	—
„ „	IV.	37·1	36·3	23·8	2·8

This, Mr. Stetefeldt states, is the cheapest drying and roasting on record in any silver-mill, and the introduction of the system of employing gas producers is making rapid progress in silver-milling. Full details of the cost of drying and roasting at Aspen on a run of 12,000 tons of ore are given in the *The Engineering and Mining Journal*, New York, of June 25th, 1892.*

When ores contain a very large percentage of sulphur : *i.e.*, are exceptionally heavy. Mechanical roasting furnaces cannot, as a rule, compete with the old-fashioned reverberatory furnace, but, with a moderate amount of sulphur, many of them give excellent results. The cost of roasting (chloridizing) in the Stetefeldt furnace is said to vary from 16s. to £1 0s. 10d., and the furnace is said to cost £625 to build. A Brückner furnace, which is a favourite in small mills, will treat from 3 to 4 tons (in exceptional cases 10 tons) in 24 hours. The cost of roasting in it varies from 10s. to £1 per ton. An improved form, with double cylinders set tandem, is estimated to roast 20 to 40 tons of refractory ore (in inverse proportion to the percentage of sulphur it carries) at a cost

* Page 660.

of £4 3s. 4d. per day, *i.e.*, 2s. 1d. to 4s. 2d. per ton. Mr. Brückner's estimate of the cost of a double-cylinder plant is £2,255 erected.

In many of the best modern mills where dry-crushing is practised the furnaces are placed at the extremity of the battery line, a little behind it, and not in front, as they used to be.

Mechanical furnaces of the improved Brückner and Howell-White type (a modification of a Hocking and Oxlands calciner), and the Stetefeldt and O'Harra are amongst those in most general use.

If the ore requires roasting, dry stamps are invariably used, the mortars being covered in with a wooden housing, to which exhaust fans are attached to draw off the dust into pockets (emptied at intervals), and the dry ore is moved by screw-conveyors or horizontal endless-belt tables to the feed-pocket of an elevator, which raises it to the hopper of the furnace if a mechanical roaster be employed.* When the ore is roasted a chlorination assay† must be made of every charge.

The floors of modern wet-crushing mills are laid slightly inclined towards a reservoir connected with the pulp-tanks, double-planked and tarred, and the mill supply of quicksilver is almost always handled mechanically by a mercury pump. It is imperative, however, that it should be of first-class make, as a poor device for handling quicksilver is more extravagant than hand labour.

The variations in the details of the plant and method of manipulation are capable of so great a number of permutations that it would be useless to attempt to go into the subject fully in this paper. The author should, however, state that the fineness to which the ore can be reduced is to no small extent determined by the capacity of the settlers to work off the coarse sands without loss of mercury.

Although it has been found by experience that some ores roast as well if crushed through a 30 as they would through a 40 mesh screen; some heavy ores, *i.e.*, those that contain a great deal of sulphur, give a low chlorination and extraction, unless crushed finer, say to a 50 mesh. The limit of coarseness to which it is ordinarily practicable to carry crushing with stamps is, the author believes, about 30 mesh.

* A new form of elevator and conveyor made by the Jeffrey Manufacturing Co., of Columbus, Ohio (a description of which is given in *The Engineering and Mining Journal*, New York, of March 4th, 1893, page 201), consisting of a steel cable, to which a number of iron diaphragms of suitable shape are attached at intervals, which travel in a trough of corresponding shape, would seem to be particularly applicable to this purpose.

† The method of doing this is excellently described by Kustel, *Roasting of Gold and Silver Ores*, second edition, 1880, page 32, *et seq.*

It cannot be too strongly emphasized that one of the most important points in pan-amalgamation is cleanliness about the works, and the use of clean quicksilver; though order and neatness, with the polish that comes of the use of elbow-grease, are factors of economy that ought to be naturally looked for in all mining plant.

Bichloride of copper is supposed to be the active agent in the Washoe process in the reduction of sulphide of silver; just as bichloride of mercury* attacks gold and amalgamates with it when ordinary quicksilver will not touch the yellow metal.

If it is worth while putting up a well-constructed mill building (which cannot be done without corresponding expense) it is worth while keeping it in first-class repair.

The cost of an ordinary 20 stamp dry-crushing plant, including rock-breaker, mechanical-drier and roasting-furnace, cooling-floors and elevators, with stamps and conveyors, and the necessary pans, settlers, sluices, etc., with a quicksilver-tank system, will not, in most cases, fall far short of £7,100, in London, exclusive of local freight and erection charges. It would weigh about 234 tons.

The cost of a 20 stamp wet-crushing silver-mill plant, with rock-breaker, automatic-feeders, stamps, pulp-tanks and pans, settlers, etc., will not generally come to less than £6,525, and weigh about 186 tons.

The cost of treatment, milling ores wet, varies from 12s. 6d. to £1 17s. 6d., employing the Washoe process. A high average being about 18s. 9d. per ton.

The cost of treatment, milling ores dry†, varies from £1 5s. to £2 11s., and when the ore requires roasting it will average from £1 13s. 4d to £3 2s. 6d., and sometimes as high as £5 16s. 8d. per ton.

The great variations in silver ores, conditions of working, and methods of extraction make it impossible to give more than very general estimates of cost, as it fluctuates frequently in the same district, with differences in price of fuel, labour, freight, chemicals, etc.

* The Designolle process.

† The writer alludes here to a modification of the Reese River process, which dispenses with roasting; as examples of which, we have the Eberhardt at White Pine, and the Lancaster mill at Tuscarora. It is practised on the grounds that a much higher percentage is saved, if the ore contain chlorides, as the finely divided horn-silver is likely to be lost if crushed wet, although the slimes are thin. It is applicable also to some ores which produce an excessive amount of slime, which escapes the settling-tanks.

A 25 stamp wet-crushing silver mill, running 24 hours, generally needs the following crew of men :—

2 rock-breakermen.
2 battery feeders.
2 amalgamators.
2 engineers (and in some cases 2 firemen).
1 mechanic.
1 foreman and assayer.
2 assistant amalgamators.
3 tank-men.

The Grand Prize* (a 20 stamp dry-crushing and chloridizing mill) employs :—

No. of Men.		Wages per 12 Hours' Shift. £ s. d.
2	amalgamators	at 1 0 10
2	„ helpers	„ 0 16 8
2	chloridizers	„ 1 0 10
2	„ helpers	„ 0 16 8
2	battery feeders (tenders)	„ 1 0 10
2	engineers (drivers)	„ 1 0 10
4	firemen	„ 0 16 8
1	melter and retorter	„ 0 16 8
6	dry-kilnmen	„ 0 16 8
1	blacksmith	„ 1 0 10
4	labourers	„ 0 16 8
28		

The Lancaster* (a 10 stamp dry-crushing raw-amalgamating mill) employs :—

No. of Men.		Wages per 12 Hours Shift. £ s. d.
2	amalgamators	at 1 0 10
2	„ helpers	„ 0 16 8
2	battery tenders	„ 0 16 8
2	engine drivers	„ 1 0 10
3	firemen	„ 0 16 8
2	dry-kilnmen	„ 0 16 8
3	labourers	„ 0 16 8
16		

Examples of the Washoe Process.

The ores of Mineral Hill, Nevada, consist of chloride of silver, bromide of silver, argentite, polybasite, stephanite, carbonate and molybdate of lead, carbonate of copper, and some manganese, occurring in a limestone-formation in irregular deposits. Mr. Eissler† states that, being of a complex character, they were originally treated by roasting milling, but he

* Egleston, *Metallurgy of Silver*, page 437. † *The Metallurgy of Silver*, page 155.

subsequently worked them successfully on the Washoe principle by dry-crushing and amalgamation (the modification of the Reese River process, before alluded to).

A point of special interest is the presence of bin-oxide of manganese in the ore. This mineral appears to have a deleterious effect on the amalgamation, its presence being indicated in the settlers by a thick froth, which in spite of dilution with water, carried off flowered quicksilver. The charge, Mr. Eissler states, which gave the best results, was found to be 1,500 lbs. of ore mixed in the pans with 15 to 20 lbs. of salt and 3 to 5 lbs. of bluestone. Treating 18 tons per diem, the cost was as follows:—

	£	s.	d.
Superintendent, who also acted as assayer	2	1	8
Master mechanic	1	5	0
Carpenter	1	5	0
Two engineers at £1 0s. 10d., 1 day and 1 night ...	2	1	8
Two men tending rock-breaker at 16s. 8d.	1	13	4
One man at dry-kiln, and to take battery samples, 1 day and 1 night	1	13	4
Two battery-feeders, 1 day and 1 night at 18s. 9d. ...	1	17	6
Two pan-men during the day	1	13	4
Two pan-men and one retorter at night	2	10	0
400 lbs. of salt at 3d.	5	0	0
8 cords of wood at £1 5s.	10	0	0
Loss of mercury, 30 lbs. at 5s.	7	10	0
Wear and tear of iron, and repairs	4	3	4
Oil and incidentals, sulphate of copper and assay materials	3	2	6
	£45	16	8

Cost per ton, £2 10s. 11d.

The ores of Pioche, Lincoln County, Nevada, which contain on the average 3 to 5 per cent. of lead (cerussite and galena) have been treated successfully by the ordinary Washoe process, the ore being worked up to over 82 per cent.; when assaying £27 1s. 8d. per ton and yielding bullion, the average fineness of which was somewhat below 700, containing lead and some copper.

To extract the greater part of the lead, the quicksilver and amalgam after leaving the settlers was strained in sacks suspended in a large box filled with water, heated with steam by a ½ inch pipe. Lead amalgam at the temperature of boiling water remains liquid, and will therefore strain through with the excess of quicksilver. As a certain amount of silver and copper amalgam also passes through, the mercury is run into a smaller box cooled with water, and when cold strained in the usual way,

leaving an amalgam of lead containing a small amount of the other metals. This lead amalgam, when retorted, gave bullion containing 6 to 20 per cent. of silver, very little copper, and only a trace of gold. The amalgam in the first sacks gave bullion from 550 to 680 fine in silver, and finer in inverse proportion to the amount of copper in the ore.

The amalgam from the second straining of the quicksilver, with ore of normal character, gave bullion 60 to 200 fine. When the ore was amalgamated without chemicals the bullion was 300 to 350 fine, and when amalgamated with salt and bluestone, but not strained in hot water, 400 to 450 fine. The ore contains on an average £1 0s. 10d. in gold to over £20 16s. 8d. worth of silver, this proportion remaining very constant. 45 to 55 per cent. of this gold is extracted. The bullion contains 0·0003 to 0·0015 parts of gold, and occasionally as high as 0·0030 parts. With bullion 500 to 600 fine (after passing through the hot-water straining process) the loss of quicksilver amounted to $4\frac{3}{4}$ lbs. per ton of ore, due probably to the formation of chloride of lead and the subsequent formation of sub-chloride of mercury. Another source of loss was the formation of floured lead-amalgam, which had the dull appearance of lead and floated off in flakes on the surface of the water. The chloride of copper formed rapidly destroyed the castings of the pans. The proportion of retorted bullion to amalgam in working the Comstock, White Pine, and Idaho ores is as 1 to $5\frac{1}{2}$ to 6; in amalgam containing a large amount of copper as 1 to 7 to $7\frac{1}{2}$; and in very base amalgam as 1 to 4.*

The ores of the Silver Reef in the Harrisburg district, Utah, are of a remarkable character, consisting of chocolate-coloured sandstone with fine chloride of silver disseminated through its mass, and where organic remains (such as leaves and stems of trees) are found embedded in it, the silver is present in a pure metallic state. In the Stormont mine, the ore is found in a zone 10 to 100 feet thick, often associated with fossil-remains, etc., and is bounded by red sandstones above and white sandstone below. The ore is very easily crushed and disintegrated, a 750 lbs. stamp putting through 7 to 8 tons per 24 hours' crushing through a 40 mesh screen, so that 5 to 10 stamps will more than supply 12 pans, working $1\frac{1}{2}$ tons pan charges. Considering the size of the mills, the cost of milling is in consequence extremely low. The ore averages £6 5s. per ton. The tailings vary greatly in richness, according to the character of the ore. With sandstone ore, they will often carry 12s. 6d. per ton, while with shaly ore they may run £2 1s. 8d. or more.

* Eissler, *Metallurgy of Silver*, page 133.

Mr. R. P. Rothwell gives the cost of a year's working at three of the mills of the district as follows:—

Per ton of 2,000 lbs.	Christy M. and Mg. Co. 14,249 tons. £ s. d.		Stormont Co. 9,983 tons. £ s. d.		Leeds Co. In 1878 12,064 tons. £ s. d.	In 1879 4,679 tons. £ s. d.
Labour & salaries	0 11 10½		0 12 4½		0 9 2	—
Bluestone ..	(2·1 lbs.) 0 1 3½		(1·¾ lbs.) 0 1 1			
Mercury ...	(1·22 lbs.) 0 2 5		(1·13 lbs.) 0 2 4½			
Salt	(25·8 lbs.) 0 2 1½		(20·00 lbs.) 0 1 2¾		0 13 5	—
Fuel	0 5 5½		0 1 10¾			
General supplies	0 3 7½		0 1 10½			
Incidentals ...	0 1 8½		0 0 6			
	£1 8 6		£1 1 4		£1 2 7	£0 17 2
Hauling ore to mill	0 3 0½		0 8 4		0 1 4	0 1 0½

In parts of Mexico, silver ore which falls below £6 5s. per ton is not available for the Washoe or patio processes, owing to the excessive cost of transport, and fuel. At Guanajuato, for example, packing on mules costs 14s. 7d., and treatment of the ore £2 7s. 11d., whilst mining, pumping, hoisting, etc., add a further charge of £2 10s. Wood costs £2 1s. 8d. and coal £4 11s. 8d. per ton. The district, however, is said to be a rich one, one group of mines north of the city of Guanajuato being credited with a production of 312,860,000 dollars between 1548 and 1889, out of a total of 650,000,000 dollars worth of silver obtained from the district.

Mr. W. L. Austin gives* the subjoined particulars of the cost of running a wet-crushing silver-mill in the Tombstone district, stating however that, owing to the arrangement of the plant, which works under some disadvantages consequent upon its alteration from dry-chloridizing to wet-crushing and other reasons, a reduction of 20 per cent. in the cost of milling could be effected under more favourable circumstances.

A novel feature to be noticed is that the shoots leading from the ore-breaker to the bins are not only provided with the ordinary grizzleys, the bars of which are set $\frac{3}{8}$ inch apart, but the bottom of the shoot itself is fitted with a shaking-frame covered with screens of the same mesh as those used in the battery.

This relieves the batteries materially, and decreases the amount of slimes, increasing the capacity of the mill 5 per cent. or more, depending on the fineness of the ore and its percentage of moisture, as the ore fine enough to pass the screens, goes direct to the pans without passing through the battery at all.

* "Silver Mining in Arizona." *Trans. Am. Inst. Min. Eng.*, vol. xi., page 91.

The average gross weight of the stamps is between 750 and 800 lbs. The shoes, weighing 120 lbs., have an average life of one month, when worn down to about 35 lbs.

Hendy Challenge feeders are used, and the stamps crushed 2·9 tons of medium hard rock in 24 hours for the first six months they were in operation, using a 30 mesh screen.

After 4 hours' grinding in the pans, the average extraction was found to be about 81·04 per cent., and nothing material was gained by prolonging the treatment, but by the use of salt and bluestone 87 per cent. could be got out of the ore, which contained only 7 per cent. of its silver as chloride.

So long, however, as coarse crushing was adhered to in the battery, the bullion was much debased. The remedy was found to be to crush finer, using 35 mesh screens and amalgamating without grinding in the pans, by which means the bullion was kept at ·970 fine without sacrificing the milling percentage. The ore contains about 8s. 4d. in gold, and 43 per cent. of this was also recovered. It may be mentioned that the debasing of the bullion was chiefly due to lead, which lowered it down to ·200 and ·300, and whenever wulfenite appeared in the ore this was remarked to a much greater extent than when the lead was in the form of cerussite or galena.

The average cost of milling for five months was with 20 stamps :—

	£	s.	d.
Labour	0	10	6
Fuel...	0	4	4½
Chemicals and mercury	0	3	2½
Lubrication	0	0	2
Illumination	0	0	1½
Castings	0	1	4½
Supplies	0	0	8
Per ton	£1	0	5

The cost of labour (subdivided) treating 1,730 tons, based on one month's work was estimated to be :—

	s.	d.
Crushing	2	2
Amalgamating	2	9½
Power-pumps, etc.	1	11½
Foremen, etc.	3	7½
Tailings-pits	0	5½
Per ton	11	0

The loss of mercury averaged about 5s. 5d. per ton, and about 0·11 cords of wood,* and 1,200 gallons of water, were consumed per ton milled.

Mr. Austin also gives another instance of the cost of wet-crushing in Arizona, estimated on 2,643 tons crushed in a 20-stamp battery, furnished with a rock-breaker and automatic feeders, separating the coarse sands in hydraulic sizing-tanks, and working each sand-class by itself, for which the ore alluded to was particularly suitable, being entirely free from base metals with a gangue of light specific gravity.

The stamps weighed 800 lbs., and put through on an average 5 tons per head in 24 hours.

Cyanide and lime, in the proportion of 14 lbs. of the former chemical and 120 lbs. of the latter per 100 tons of ore, were used in the pan treatment. The mercury was pumped through the mill for distribution.

The motive power was a 200 horse-power horizontal engine (42 inches by 20 inches cylinders, run at 60 revolutions per minute), and the boilers, which were tubular, 15 feet 6 inches by 54 inches, carrying 85 lbs. steam pressure, burnt 16 cords of wood irrespective of the boiler for the pump, which burnt 8 cords per week. The cost per ton of ore was:—

	£	s.	d.
Labour	0	5	1½
Supplies	0	7	7
Assaying	0	4	5½
	0	17	2

The cost for labour as given above is thus subdivided:—

	£	s.	d.
Crushing	0	1	1
Amalgamation	0	0	10
Power, pumps, and repairs	0	1	8
Foreman, melter, etc.	0	1	6½
	0	5	1½

The cost of material as given above is thus subdivided:—

	£	s.	d.
Mercury	0	1	9
Chemicals	0	0	3½
Castings	0	1	2½
Illumination and lubrication	0	0	3½
Fuel, including pump	0	3	3
Supplies	0	0	9½
	0	7	7

* This includes pumping the water supply 200 yards (with a lift of 100 feet vertical); 7 cords of mixed wood (black oak, white willow, and pine), costing £1 17s. 6d. per cord, were used on the average per day. The mill-engine was provided with a Meyer cut-off.

The consumption of wood per ton of ore was 0·15 cords, and of mercury 0·96 lbs.

The bullion averaged ·995 fine.

Stated in gallons, the quantity of water used per boiler in a silver-mill is 7½ gallons per horse-power per hour. For each stamp, 72 gallons per hour; for each pan, 120 gallons per hour; and for each settler, 60 gallons per hour.

The consumption of wood (not including that used for roasting) in a dry-crushing mill is put by Prof. Egleston at about ⅓ cord per ton.

The power required for a Brückner roasting-cylinder is estimated at about 2 horse-power, and for the Howell-White 1½ horse-power.

The ore according to its baseness loses 3 to 15 per cent. by weight in roasting.

The value of stock on hand necessary to run a dry or wet-crushing silver-mill, including wood, mercury, castings, chemicals, etc., is very variable.

Examples of Roasting Milling Treatment.

The ore of the Ontario mine is a very base one, being composed of zinc, lead, and silver sulphides and silver chloride in a quartz gangue. This becomes baser with the increasing depth of the workings. The bullion obtained runs about 600 fine and contains no gold. The average grade of the ore treated is 100 to 130 dollars per ton, and the amount treated varies from 50 to 55 tons per day, though formerly (when it was less base) 65 tons were handled daily. The ore is roasted in Stetefeldt furnaces with the addition of about 15 per cent. (dry weight) of salt.

After leaving the furnace the ore goes to the cooling-floor, where it remains piled up for 18 hours. This increases the chlorination from 3 to 8 per cent. After being damped it is run to the pan-room. Each pan-charge of ore weighs 2,500 lbs, to which 1 per cent. of salt is added, and the pulp made up with the addition of hot water to proper consistency. The muller makes about 65 revolutions per minute, and is held 1 inch above the pan-bottom so that it does not grind. About 1 lb. of zinc and 300 lbs. of mercury are added to the pan after it has run 1 hour (and is hot), and amalgamation is continued for 7 hours. From the pans the pulp is drawn into settlers, which run 4 hours, making 40 revolutions per minute, and after running for 1 hour cold water is let in, and the overflow discharging the tailings is set running. The mill is worked up to from 88 to 92 per cent. of the value of the ore, that being counted as the amount chlorinated. The tailings carry off 8 to 12 per cent. of the silver.

The cost of treatment on a production of 50 tons per day was stated as follows* :—

No. of Men.	Occupation.		Per Day. Dollars.	Per Ton. £ s. d.
1	Foreman...	...	10·00	0 0 10
1	Assayer ...	...	6·00	0 0 6
3	Machinists	at $4·00	12·00	0 1 0
2	Carpenters	at 4·00	8·00	0 0 8
2	Blacksmiths	at 4·00	8·00	0 0 8
2	Engineers	at 4·00	8·00	0 0 8
2	Foremen	at 3·50	7·00	0 0 7
9	Dry-floormen	at 3·50	31·50	0 2 7½
3	Batterymen	at 4·00	12·00	0 1 0
6	Roasters	at 4·00	24·00	0 2 0
12	Cooling-floormen	at 4·00	48·00	0 4 0
4	Carmen	at 4·00	16·00	0 1 4
4	Amalgamators	at 4·50	18·00	0 1 6
1	Retorter	at 4·00	8·00	0 0 8
1	Melter	at 4·00		
4	Labourers	at 2·50	10·00	0 0 10
4	Watchmen	at 3·00	12·00	0 1 0
2	Ore-floormen	at 3·50	7·00	0 0 7½
3	Clerks	at 4·00	12·00	0 1 0
66			$257·50	£1 1 6

Supplies.		Per Day. Dollars.	Per Ton. £ s. d.
Salt, 10 tons	at $8·00	80·00	0 6 8
Quicksilver, 175 lbs.	at 0·50	87·50	0 7 3½
Wood, 15 cords	at 4·50	67·50	0 14 8½
Coal, 12 tons	at 8·25	99·00	
Castings			0 6 3
Oil and waste...			0 1 0½
Sundries, chemicals, etc.			0 2 1
Hauling from mine ...			0 2 0½
Charcoal, assaying, and melting			0 1 0½
			£2 1 1½

Total cost per ton, £3 2s. 7½d., exclusive of office expenses, general superintendence, repairs, and insurance.

The report of the Granite Mountain Company, Phillipsburg, Montana, one of the largest dry chloridizing roasting-crushing plants in America (running 153 stamps), gives the following particulars of working for the year ending July 31st, 1891 :—

Ore, tons crushed, wet,	72,623	= dry weight,	68,850
Salt,	„	„	10,807	= „	10,645
	Total		83,430		79,495

Average moisture in the ore, 5·2 per cent.; average moisture in salt, 1·5 per cent.

* R. P. Rothwell, *Trans. Am. Inst. of Min. Eng.*, vol. viii., page 557.

The salt and ore are mixed before crushing.

The ore averaged 50·59 ounces of silver, and the cost of milling was £2 1s. 8d. per dry ton, divided as follows :—

	£	s.	d.
Labour and superintendence ...	0	14	5
Salt	0	8	3¼
Fuel	0	6	10¼
Mercury	0	4	8
Castings	0	2	7½
Chemicals	0	1	2½
Water	0	0	6¾
Miscellaneous	0	3	0¾
	£2	1	8

Equal to £1 19s 5¼d. per wet ton.

The company's three mills of 20, 43, and 90 stamps respectively are known as A, B, and C, and have cost, with improvements up to date, £145,843 17s. 9½d., as shown by the trial balance sheet. The cost varied from £1 16s. 7½d. in mill C, which crushed 42,153* wet tons, to £2 11s. 10d. in mill A, which crushed 9,934† wet tons.

The saving of silver amounted to 90·7 per cent.

The report of the Elkhorn Company, of Montana, for 1891, affords another excellent illustration of close saving and business-like management.

The mill, which crushed 11,645 tons (dry) during the 12 months ending December 31st, 1891, saved 93·78 per cent‡ of the silver contents of the ore at a cost of (9·226 dollars)§ £1 18s. 5¼d., a close approximation to the figures just given.

Tailings Mills.

Silver-mill-tailings are generally concentrated on concentrators, or on blankets, in sand-sluices, and either leached, or treated in pans on the Comstock system (using chemicals); which has given rise to what are known as tailings, or auxiliary mills.

Tailings from silver mills can in fact often be treated profitably by storing them in dams, leaving time, air, and chemical agency to effect their oxidation, and then treating them raw in pans, no drying or chloridizing being required.

The chief drawbacks against this method are the uncertainty as to the time necessary for complete oxidation of the minerals present, which may be taken from 2 to 4 years, and the necessity of constructing large storage dams.

* Representing 40,067 dry tons. † Representing 9,318 dry tons.

‡ In 1890, 86·83 per cent. was saved, the increase in 1891 is attributed to using extra salt. § For analysis, see Appendix.

Mr. Eissler states* that, in some establishments of this kind in Nevada, the quantity of sulphate of copper supplied to the pans with this system of treatment varies from 3 to 6 lbs. per ton of tailings, while the salt amounts to 20 or 30 lbs. The pans are covered, and supplied with steam at a high temperature.

He places the yield at about 60 per cent., dealing with an ore worth £3 to £3 10s. per ton, and gives the current cost as follows :—

	£	s.	d.
Labour	0	6	0
Quicksilver lost	0	4	0
Salt	0	3	0
Sulphate of copper	0	2	6
Fuel	0	5	0
Castings	0	0	6
Total	£1	1	0

In Avery's tailings-mills in Washoe valley, where wood is £1 4s. per cord, the cost per ton is stated to have been 14s.

It is usual in Nevada for two sets of samples to be taken as a basis upon which to determine the value of the ore, when it is treated at customs establishments. One of these samples is taken from the waggon conveying the ore from the mine to the mill, the other from the mill after the ore has been crushed. The waggon sample is obtained by drawing from each waggonload of ore a sample of rock, and mixing the total number of samples of the loads sent to any one mill at the end of the day. This bulk sample is then quartered down, the mill being charged with the weight of ore, and the amount of gold and silver represented by assay. The mill sample, which is to serve as a check on the waggon sample, is taken by allowing the crushed ore as it comes from the battery to run into a pail, held at the end of the trough leading to the tanks.

A sample of this kind is taken in most mills every hour, in some every half-hour, and the accumulated samples are well mixed, dried, and reduced to a sufficient quantity for assay.

The pail or vessel in which the crushed ore is caught must not, of course, be allowed to overflow.

In some mills the samples are taken from the tanks after the sand has deposited itself in them, either from the surface or preferably with a tube sampler.

Strange as it may seem the waggon samples and mill samples generally differ in yield, and the former is usually the highest.

This was proved in the late celebrated suit tried before Judge Hebbard, Fox *versus* the Hale and Norcross Co. In practice, therefore, both assays are duly considered, and an adjustment arrived at.

* *The Metallurgy of Silver*, page 114.

A. D. Hodges, in an interesting paper on pan-amalgamation,* estimating on 46,500 tons of Comstock tailings treated at Drayton, Nevada, calculates the cost of treating them as follows :—

	s.	d.
Preparing and hauling	1	3½
Milling...	8	1½
General expenses	0	10½
Total per ton ...	10	3½

Some of the sand so treated seems to have run about 15s. 3½d. in silver, and 4s. 8½d. in gold, total £1, and yielded in bullion 75·8 per cent. of the silver, and 25 per cent. of the gold, total 63·8 per cent., whilst some of the slimes averaged £2 15s. 3d. silver, and 9s. 5½d. gold, total £3 4s. 8½d., and yielded 85·8 per cent. of the silver, and 42·5 per cent. of the gold, total 79 5 per cent.

The loss of quicksilver, treating sands at Drayton, ran under ½ lb., and with slimes from ¾ to 1 lb. per ton.

The bullion from the treatment of the sands did not run over 150 fine, and with the richer slimes the best results were got when it was from 200 to 250 fine.

In Dakota, it may be mentioned that with free-milling gold ores, it cost 4d. for mill labour, to produce 4s. 2d. worth of gold.

In Nevada, on the other hand, though over five times as much work is required per ton treated, it costs only about 3½d., but this seeming contradiction is explained by the higher grade of the Nevada ores.

A 20 pan mill, without any special arrangement for catching mercury working on tailings, made a gain of 700 lbs. of mercury in 6 weeks, which was but a small part of the mercury actually contained in the slimes.

In view of these losses it generally pays to employ concentrators to work up the mill-tailings. Sometimes a machine, known as a Varney amalgamator, is set in the tailings-sluice.

As battery slimes require no grinding it would seem most advantageous to charge the quicksilver direct into the pan, but experience has shown that better results are obtained by charging the chemicals with the slimes and thoroughly mixing them for about two hours with the muller up, and then adding the mercury, as the slimes form a pasty mass which might hold and carry off the finely divided quicksilver. In treating such material one-half its bulk in sand (tailings from the pans), and sometimes more, is added to the charge.

In many so-called free-milling ores of silver, or silver and gold combined, there are small quantities of sulphides of the base metals, not

* *Trans. Am. Inst. Min. Eng.*, vol. xix., page 231.

sufficient in quantity or value to make the ore suitable for roasting, yet enough to prevent a high extraction by free-milling, besides increasing the cost. Such ores can be best treated by the combined process of concentration and amalgamation.

Stamping the ore wet, passing it over copper plates, concentrating in vanners, and then amalgamating the tailings by the continuous process, lessens or dispenses with grinding in the pans, decreasing the wear of castings and fuel consumption; decreases the losses of quicksilver; increases the capacity of the battery by permitting coarser crushing; raises the percentage of extraction, and gives higher grade bullion.

The Montana Company by adopting this process obtained, it is said, an increased saving over ordinary pan treatment of from £1 13s. 4d. to £2 1s. 8d., decreased the loss of mercury to ¾ lb. per ton of ore, and increased the tenure of the bullion from ·500 to ·900 fine. The cost of the process is from 12s. 6d. to £2 1s. 8d. per ton, and less water is required than with ordinary pan-amalgamation.

It is sometimes more expedient (depending on the base metals present, and the way the ore breaks) to crush finer in the battery and grind less, as before mentioned.

Since ore will break naturally where there is most mineral, by disengaging it as far as possible in crushing, as mercury has a greater affinity for the precious than for the base metals, there is less likelihood of its becoming foul and inactive through taking up lead and other things which grinding tends to make it do.

In Arizona the loss in melting bullion averaging ·938 fine (from volatilization and skimmings) is stated by Mr. Egleston to be 7·55 per cent.

Ores, like some of those of the Comstock, that contain their gold and silver more or less free, will mill from 60 to 80 per cent. by the Washoe process and in exceptional cases, like those of White Pine (Nevada) and Silver Reef (Utah) to even 85 per cent. Roasting milling, on the other hand, gives an extraction of from 80 to 93 per cent., and in exceptional cases 94 per cent.

The combined process yields often 75 to 85 per cent.

The Patio Process.

This very ancient and interesting process cannot be passed by without comment, as Mexico, which is its home, so to speak, is, it must be recollected, the second largest silver producer in the world, and it is safe to say that three-fourths to seven-eighths of the silver it returns is obtained by the patio process.

The practice at San Dimas (State of Durango) may be considered typical, and is excellently described by Mr. R. E. Chism,* whose paper contains detailed information on the subject. Englishmen and Americans alike have gone into Spanish America from time to time expecting to revolutionize this time-honoured system by introducing more modern metallurgical methods, but in most cases they would appear to have failed in doing so, leaving abandoned reduction works all over the country as monuments of their errors. If we look for an explanation, it is to be found in the neglect of local conditions on which the success or failure of every process more or less depends.

The patio, with its cheap plant and fairly close extraction in Mexico, where saving of time is a secondary consideration, where transportation is difficult, where the ores are rich, and where animal power, space, and labour are cheap, whilst fuel and other necessaries are dear (Central America possessing a suitable climate, and labour accustomed to the work from time immemorial) can, in fact, hold its own for the treatment of ores, which do not contain more than a trace of lead and zinc, where other processes would fail.

The silver, which the patio yields, is almost free from impurities and baser metals. An assay of several bars showed an average fineness of 0·994 silver and 0·035 gold. Mr. Chism gives the cost of treatment in a large hacienda where the tahonas were in two groups, and were worked by gear connected with an over-shot waterwheel, where the breaking was done by wooden stamps shod with iron (also driven by a waterwheel), and where the washing was done by a water-power washer, working a trilla of 19 tons, as follows :—

	Cost per Ton of 2,000 lbs. £ s. d.
Breaking, grinding, and cost of tools	1 7 9
Amalgamator's wages	0 6 11
Scraping tahonas	0 0 8
Carrying and washing scrapings	0 0 5½
Concentrating tailings of scrapings	0 0 3½
Carrying slime from tahona to patio	0 1 9
Mules and keep	0 15 6
Labour, spreading trilla and mule driving	0 6 8
Labour, washing trilla	0 2 4
Charcoal, for retorting silver	0 1 11½
Concentrating tailings of trilla	0 8 7
Materials— £ s. d.	
Salt, 600 lbs. at 4d. ... ... 10 0 0 ...	0 10 6½
Sulphate of copper, 125 lbs. at 1s. 0½d. 6 10 2½ ...	0 6 10½
Precipitated copper, 25 lbs. at 2s. 9d.... 3 8 9 ...	0 3 7½
Quicksilver, 133 lbs. at 2s. 7½d. ... 17 6 4 ...	0 18 2½
Total cost	£5 12 1½

* *Trans. Am. Inst. Min. Eng.*, vol. xi., page 61.

The cost to the owners would probably not exceed £5 4s. 2d., as the above costs include a certain charge for profit.

At works where the tortas are of small size (about 10 tons) and the ore is broken without the aid of water-power, inclusive of superintendence and interest on plant, the cost is said to be £5 14s. 11d.

The concentrates obtained from the planilla-treatment and washing in the boliche are shipped and smelted in Germany.

From 70 to 75 per cent., and in some few cases perhaps 80 per cent., of the assay value of the ore in silver is saved by the patio process, 72 per cent. being probably about the average; whilst of the gold present 40 per cent. is lost, 20 per cent. of the remainder goes with the silver, and the rest is recovered from the tailings or is caught in the tahona.

The process from start to finish takes from 23 days to 7 weeks.

In *The Engineering and Mining Journal*, New York, of May 7th, 1892,* Mr. E. du B. Lukis describes, what he claims to be, a recent improvement on the patio system specially applicable to ores containing antimonial and arsenical sulphides of silver, together with ordinary sulphides and some chlorides.

The ore, instead of going direct to the patio after crushing, is first roasted for 10, 12, or 20 minutes. The object of this roasting is merely to give a quick start to the process, and while, owing to its rapidity, it is said not to entail a loss of over 1½ to 2½ per cent. of the silver by volatilization; comparative trials have, it is asserted, proved the benefit to be derived therefrom.

It is not desirable to roast sweet, but only to break up the molecular affinity of the particles of the mineral by heat, so as to hasten the progress of the after-chemical process with a view to save labour and time.

The most important part of the modification in treatment, however, lies in the use of hyposulphite of soda, for if roasted ore be treated by the ordinary patio method it will not work, the quicksilver gets hot at once, and nothing can be done with it.

It is also claimed that it is now possible to arrive at the quantity of sulphate of copper that will be required to beneficiate the cake from the start, reducing the treatment nearly to a certainty. To avoid the usual loss of quicksilver, the amalgamation is stopped before the difficulty of the patio treatment commences (when in fact the free silver ores have been amalgamated), the amalgam is then washed out in the usual way, and the more refractory silver minerals are concentrated. The results are stated to have been satisfactory, showing an extraction of 92 per cent. of the assay value.

* Page 496.

The ores taken for trial were of a mixed class. One-third of the silver being antimonial or ruby silver, the remainder mostly silver glance, but with pyritiferous silver ore and some chlorides. The assay value was 50 dollars per ton. The heavy minerals amounted to about 10 or 15 per cent. of the whole.

The ores were stamped dry to pass a 60 mesh screen, and roasted in an ordinary reverberatory furnace with a hearth measuring 8 feet by 6 feet. The temperature was raised to a good red heat. Charges of 400 lbs. were mixed with 2½ per cent. of salt, spread over the hearth and kept gently stirred with a rabble. Eight minutes after charging, a sample was taken out and washed in the tentadura or assay horn to show the colour of the concentrates. It is by the colour that the calciner knows when to stop roasting. When sufficiently roasted, which does not in any case take more than 20 minutes, the charge is quickly withdrawn, and a fresh one introduced.

The hot ore is allowed to remain in a pile until next day, when it is spread on the patio, moistened with water, and trodden into a soft pulp, with an addition of ½ per cent. of salt. This being done in the morning, sulphate of copper can be added three or four hours later. The required quantity can only be ascertained by experiment, but in the case cited 4¾ ounces per 25 lbs. were found the right quantity. After this addition the pulp is again trodden and left till next day.

Early next morning a guide is taken from the cake by a peoñ who walks across it in two diagonal lines taking small portions of the pulp here and there, and accumulating a sample as near as can be guessed of about 100 lbs. This is placed in a corner of the patio, and divided into four portions of 25 lbs. each, to each of which different quantities of sulphate of copper are added, say 1, 2, and 4 grammes (15, 30, and 60 grains) respectively. Assays of each are taken in turn (commencing with the pulp of the cake itself), and mixed up with a globule of mercury in the tentadura and washed, when the action of the sulphate on the mercury will be seen by its colour. It ought to remain bright and quick, and sulphate can be added in small quantities to the guide until the mercury shows a trace of heat, *i.e.*, becomes discoloured and leaden in tone, which indicates that a gramme (15 grains) too much per 25 lbs. has been added.

The amalgamator now knows that he may add 1, 3, or 5 grammes (15, 45, or 75 grains) as the case may be per 25 lbs. of cake. Done with care and patience, this should be a sure guide as to the total quantity of sulphate the cake can stand.

If the process were now continued, as with raw ores, the mercury would become very hot by next day, and soon not work at all, that is to say become dirty in appearance and cease to form silver amalgam. To avoid this trouble, hyposulphite of soda has been used and found to work successfully, whilst it helps to hasten the process. It must, however, be employed with great care, the quantity being found by trial for determined mixtures of ore, and upon this depends its success. Roughly speaking, ½ ounce per 100 lbs. of pulp will be sufficient, more would destroy the mercury.

Having decided upon the additional quantity of sulphate of copper that can be added on this the second day, and having trodden it into the pulp thoroughly, the hyposulphite of soda may be added and trodden in the same way, and immediately afterwards the mercury should be sprinkled over the cake and trodden in, in such proportion as to take up by amalgamation two-thirds of the assay value, or 4 ounces of mercury per ounce of silver present.

The following or second day after incorporating the mercury, treading with horses, spading over, and taking assays will be all that is needed, unless it is found that a guide of 25 lbs. can stand an addition of sulphate of copper. This should not be necessary if the quantity be properly determined before adding the hyposulphite. More than half the mercury will be converted into amalgam by the evening.

The following or third day after incorporating, the same work is required, paying more attention to the assays, and if necessary adding more sulphate of copper, and by evening more than three-quarters of the mercury will be taken up.

The fourth day, the treading, etc., go on, and by evening the mercury should all be converted into silver amalgam, and should be bright and dry, and free from a straw-colour that indicates loss. The next morning the baño or bath is added, consisting of 1½ ounces of mercury per ounce of silver in the cake, which collects the fine amalgam.

This has to be done quickly, and the washing of the pulp follows immediately. The amalgamd, is collecte pressed, retorted, and the bullion in due course melted down.

This treatment extracts that portion of the silver which cannot be concentrated, and it is found easy to concentrate the remainder on vanners, or other machines, obtaining, if necessary, two grades, the first assaying 1,500 ounces or over per ton, and the second 40 to 60 ounces. The waste is then too poor to treat further.

These trials were made with an average temperature of 70 degs. Fahr. in the shade.

The loss of mercury, including the consumido and mechanical loss is said to be about 18 ounces for every 16 ounces of silver, as compared with 22 to 24 ounces lost in the ordinary patio. The gold is collected, if present, partly in the amalgam and partly in the high-grade concentrates. The cost of working a ton of ore assaying 50 dollars is estimated at £1 5s. 10d., extracting 85 per cent. of the silver.

The Fondo and Tina Processes.

In South America, the patio process seems to have been used up to about 1830, when, in consequence of the large quantities of negros or sulphide ores, which began to be found there, it was superseded by the fondo or calderon method, with variations known as the tina, Francke-tina, etc.* The original fondo process was invented, it is believed, in Chili, in the year 1609, by a priest, Albaro Alonzo Barba,† and was used for rich surface-ores, chlorides, bromides, and oxides, which, if not rich enough, required to be concentrated in the planilla, but it was on the whole a very slow process.

The tina process in a modified form‡ is applicable to all ores of silver except argentiferous sulphides of copper, galena, or blende, and to ores which contain more than 1 per cent. of free arsenic, which causes great losses of mercury. The inventor is not known, but it has been in constant use about Copiapo since 1862.

Prof. Egleston§ says of it:—"The whole operation is very simple, quicker with much less loss than the barrel, more certain in its reactions than the patio, and is applicable to almost all the ores found in Chili. It is even cheaper under some circumstances than the lead-fusion."

The cost of treating a ton of £8 6s. 8d. ore, not including interest on sinking fund, working 8 tons a day, is about £1 8s. 7½d.

The process now used in Bolivia, a modification of the tina and fondo processes, seems to be an advance on the former methods, as it includes many of the best points of the tina and pan processes, and may be used for base as well as docile ores of a great variety of yield.

It is closely related to the Francke-tina method, which has been described by Mr. Edgar P. Rathbone‖ and Mr. Arthur F. Wendt.¶

Mr. Wendt says of the Bolivian method, "That large losses of silver were experienced through volatilization when the ore was roasted with

* Egleston, *Metallurgy of Silver*, page 312.

† Percy, *Metallurgy of Gold and Silver*, part I., page 656.

‡ *Revue Universelle des Mines*, series 1, vol. xxxi., page 493.

§ *Metallurgy of Silver*, page 323.

‖ *Proc. Inst. Mech. Eng.*, 1884, page 257.

¶ *Trans. Am. Inst. Min. Eng.*, vol. xix., page 74.

salt in revolving mechanical furnaces, and these could only be obviated by an oxidizing roasting in reverberatories (following lump roasting in kilns), adding the salt after complete oxidation has taken place."

The extraction remains practically the same, whether the chloridization of the ores is 20 or 90 per cent. This is so entirely different from the generally conceived notion of silver-milling in the United States that too much emphasis cannot be laid on this point.

All classes of ores can be worked by the process, not even excepting galena, although the extraction, with the latter and when blende is present, is of course reduced. The bullion obtained is generally ·900 fine or over, alloyed principally with copper. Treating crude ore which carries 75 to 80 ozs. the tailings average about 10 ozs.

The use of bluestone in pan-amalgamation with the object of producing a sub-chloride of copper, which is an active agent in the pan process, has been commented upon before, but when an iron pan is used this sub-chloride is being constantly destroyed. When a copper or bronze vessel, however (like the tina), is used, the sub-chloride is, on the other hand, being constantly regenerated, and this is the essential difference between the two methods, although of course the same results can be obtained at the expense of extra bluestone in the pan in dealing with base ores.

The choice between the tina and the iron pan process must, therefore, Mr. Wendt thinks, depend on the relative price of sulphate of copper and iron-castings as compared with copper, to which must be added the extra refining charges, carriage, etc., on more or less coppery bullion.

The cost at the Real Ingenio Potosi is stated to be 90 Bolivians (£4 13s. 9d.) per cajon of 5,000 lbs., or about £1 17s. 6d. per ton, treating 8 tons per day, but this is, of course, no criterion of what could be done with cheaper freight and better labour in works of large capacity.

The process gives an extraction of 80 to 85 per cent. in Bolivia, and probably on rich chloride ores like those of Caracoles (worked by the Kroehnke process) it would yield returns running up to 92 to 96 per cent., as the extraction on roasted ores averages 90 per cent.

The Leaching or Lixiviation Process.

This process consists in first roasting the ore with salt to convert the silver into chloride, then dissolving the chloride of silver in a solution of hyposulphite of soda and precipitating it with sulphide of lime or soda (as a sulphide of silver) and refining this latter product. The Russel process is a modification of this, consisting in the use of what is known

as extra solution (prepared by adding a certain percentage of sulphate of copper to the ordinary hyposulphite solution) after the ordinary solution has extracted all the silver it will take out.

Lower cost of plant and in some cases less expense in treatment are advantages which are claimed for leaching treatment, but against these must be set the difficulty of obtaining men with practical experience and chemical knowledge necessary to conduct the process, and the fact that the reactions involved are often obscure, and sudden disturbances in working, may be introduced by changes in the character of the ore.

Since lixiviation was introduced into Mexico by Mr. Ottokar Hofmann, in 1868, it has become extensively used, and has risen to a position of importance in some parts of America for the treatment of refractory silver ores. All silver ores are capable of being treated by it with a certain amount of success, except those which contain so much lead that they are classed as smelting ore, or which on account of a clayey-calcareous or talcose gangue, do not permit of free filtration. This latter objection can, however, be overcome, if other circumstances admit of it, by a previous slight concentration. The ore must first be pulverized to fit it for the process, and though this is purely a mechanical operation it does not follow that the machine which will crush the largest amount of ore at the least cost is necessarily the best to select. One has to consider as well in this connexion, the physical condition in which the ore leaves the machine, as on this depends to a great extent a most important part of the process—the roasting. Ores which contain a considerable amount of argentiferous galena or zinc-blende require to be pulverized very fine in order to ensure a satisfactory chlorination, while a coarser pyritic-pulp (whether consisting of iron or copper pyrites) will chloridize well. In selecting machinery for the purpose attention must therefore be paid (as in every other process) to the nature of the ore.

The choice of crushing machinery, for an ore requiring leaching, lies mainly between rolls and stamps. The former produce a much more uniform grain than the latter (the ore and gangue-particles being of more even size), which is of distinct advantage for concentration, but may be the exact opposite for roasting. The most suitable condition of the pulp for roasting in dealing with ore of the kind described, is, in fact, a mixture of fine ore-particles in a coarse gangue, a condition of pulp which is promoted by dry-crushing with stamps. This is explained by the fact that the ore, as a rule, having a higher specific gravity than the gangue cannot so readily escape comminution as the lighter particles of rock, and is only discharged from the screen when reduced to a condi-

tion of comparatively fine pulp, while the lighter gangue is expelled when much coarser.

Roasting.

In roasting, the ore undergoes a chemical decomposition, and this can frequently be better accomplished if the ore-particles are of fine size. The fine condition of the pulp does not seriously interfere with the subsequent lixiviation, because most ore after a chloridizing-roasting (chlorination) becomes sandy, and will filter freely if the gangue is not of a clayey nature. In order to convert the silver into chloride, as before mentioned, the ore must be roasted with salt; and to roast successfully, a sufficient percentage of sulphides must be present, as they produce by their decomposition sulphuric acid (which acts on the salt and liberates the chlorine necessary for the formation of silver chloride). The base metals in the ore are converted into oxides, chlorides, and sulphates. The requisite amount of salt (depending on the character of the ore) varies from 4 to 10 per cent. The salt is either added to the ore before it enters the furnace, or after it has reached a certain stage of oxidizing-roasting. The selection of the proper roasting-furnace to employ is a matter of great importance: this was proved by Mr. Hofmann in Mexico, in experiments made on the San Francisco del Oro ore, in 1888.

It is to be remarked in favour of the Stetefeldt roasting furnace that it requires less salt than any other for chloridizing-roasting silver ores: the decomposition of the salt is very perfect, the chlorine and chloridizing gases emanating from the roasted ore at the bottom of the shaft, acting on the falling ore. Ores which are free from lime and magnesia, can be chloridized in this furnace with a minimum of salt, and it seems to be specially adapted to roasting ores containing copper. Mr. Aaron states that at the Surprise Valley mill, California, an average chlorination of 92 to 93 per cent. was obtained during a nine months' run, roasting silver ores of £15 12s. 6d. assay value, with only 2½ to 3 per cent. of salt. Generally, 5 to 8 per cent. of salt is required in the Stetefeldt furnace, if the ore carries lime and magnesia or a larger percentage of base sulphides. The chlorination varies with the character of the ore, and the attention with which the furnace is managed; results as high as 97 per cent. have been obtained, while they generally range from 87 to 93 per cent. Ores free from sulphur, or with only a slight percentage, should be mixed with 1 or 2 per cent. of iron pyrites, otherwise the salt cannot be decomposed. Oxidized ores carrying peroxides of manganese and iron, which give off oxygen, can be successfully chloridized by themselves. The best results are obtained by mixing oxidized with sulphide ores, more parti-

cularly if the former contain peroxide of manganese. The presence of copper is very favourable for the chlorination of the silver, and if the ore is of such a character that it will bear a high heat without sintering, the chloride of copper formed in the upper part of the shaft can be almost entirely decomposed, and very fine bullion produced by amalgamation. At the Surprise Valley mill, for example, the ore roasted at a low temperature gave bullion only 300 to 400 fine by amalgamation, the base metal being copper. By roasting at a high temperature the bullion was almost freed from copper, its average fineness being 980, running for nine months. All antimonial ores are chloridized with great facility, and with a good system of dust-chambers, the loss of silver by roasting is hardly perceptible. The same is the case with zinc-blende. In roasting ores entirely free from, or with a small percentage of, sulphides, the want of sulphuric acid must, as before remarked, be remedied by adding another substance. A cheap substitute for pyrites is found in green vitriol (sulphate of iron) of which 1½ to 3 per cent. is added when 8 to 10 per cent. of salt is used. The copper is first calcined to drive off its water of crystallization by a gentle heat, and the above percentage of the calcined material is taken. The sulphate then acts on the salt as if it were created in roasting; copperas may also be added to arsenical ores free from sulphurets. If there is a great deal of lime in the ore, it takes up sulphuric acid, forming sulphate of lime, which remains undecomposed; ores containing lime require, therefore, a larger proportion of copperas or iron pyrites, sufficient to transform all the lime into sulphate. At the same time, lime assists in decomposing the base metal chlorides in roasting (5 to 6 per cent. being sometimes added to base ores in a pulverized condition), whilst it does not attack the silver chloride; but too much must not be added if the ore is to be amalgamated afterwards. The lime should be added towards the end of the roasting, introducing 2 per cent. to commence with, and well mixing it with the ore.

Usually, ores containing not more than 8 per cent. of sulphur roast well in the Stetefeldt furnace, but in operating on the San Francisco del Oro ore the results turned out unsatisfactory. This ore contained 25·5 per cent. of zinc, 11·56 per cent. of lead, and 21 per cent. of sulphur, and proved too refractory for the Stetefeldt process. The ore (when sifted into the furnace) created by the sudden combustion of the sulphides an extremely high temperature in the upper portion of the shaft, which caused the suspended ore-particles to slag to globules (a coating of silicates being formed immediately around them), which prevented their further oxidation and chlorination. An analysis showed that 8·48 per cent. of the sulphur, in

fact, remained unoxidized. It was found also that a separation of the ore took place in the furnace, the shaft receiving the portion carrying most of the lead, while the lighter portion, containing most of the iron pyrites, was carried over into the descending-flue, where, owing to this circumstance, combined with the high temperature, that particular portion of it was better chloridized than the bulk which passed through the shaft. An excess of salt lowered the chlorination, and 12 per cent. was found to give the best results (60·8 per cent. chlorination) in the descending-flue. Another bad feature was the formation of lumps and crusts in the upper region of the shaft, and choked the lower side of the feeding-screen.

An ore of the class operated upon requires to be submitted to a long and gradually increasing temperature before the salt is added ; and as the principle of the Stetefeldt furnace is just the reverse of this, the results could not but be unsatisfactory. Heavily sulphide ores, especially if they carry zinc, require more draught in roasting than lighter sulphide ores ; and this is particularly the case when they are exposed for such a short time, as in the Stetefeldt furnace, to the action of air and heat. Mr. Hofmann's experiments in roasting the ore of the San Francisco del Oro mine in a Stetefeldt furnace go to show :—

1. An incomplete oxidation of the sulphide minerals, the main portion of the ore still containing 8·48 per cent. unoxidized sulphur when roasted with salt, and 7·6 per cent. when roasted without salt.
2. An insufficient chlorination of the silver. The highest chlorination of the ore in the shaft was only 16·9 per cent., and as 62·5 per cent. of the whole volume dropped into the shaft, the somewhat higher results obtained in the descending-flue could not much improve the average.
3. That the principle of the Stetefeldt furnace is contrary to the conditions, the maintenance of which are so essential to roasting ores containing much zinc blende and galena.
4. That, on acount of the sudden exposure of the raw ore-particles to such a high temperature, they melt to minute globules, which make the ore unfit for further treatment.
5. That a concentration of the lead minerals takes place in the shaft, which is disadvantageous.
6. That about 25 per cent. of the ore, when passing through the furnace, is changed into hard lumps of almost raw ore.
7. That the lower side of the feed-screen becomes rapidly encrusted and the holes obstructed, requiring frequent changes of screens.

The ore after passing through the shaft of the furnace had a dark, almost black colour, and continued to emit volumes of sulphurous acid gas when discharged, and for some days after, but no chlorine. Mr. Hofmann next tried re-roasting the partly roasted ore from the Stetefeldt in a modified Howell furnace. Having previously ascertained that this ore contained only 1·3 per cent. of salt, 6 to 8 per cent. more was added, and the feed regulated so as to put through from 5 to 9 tons per 24 hours. The average of thirty-three charges worked thus gave:—average of re-roasted ore, 31·42 ounces of silver per ton; average of leach-tailings, 17·55 ounces per ton; average of chlorination, 44·2 per cent.

It is a point to note that the consumption of wood in re-roasting proved to be much greater than in roasting raw ore. This is accounted for by the main portion of the sulphur, combined with the pyrites, having been burnt off in the Stetefeldt furnace. During the second roasting, the furnace was deprived of the heat of combustion derived from this source, so that additional extraneous artificial heat had to be furnished. To re-roast in this manner 8·3 tons, it took 26 cargas of wood (12 cargas = 1 cord), while it took only 16 cargas to roast 10 to 11 tons of raw ore. The roasting capacity of the furnace was also diminished. By dumping the red-hot ore from the Stetefeldt direct into a reverberatory hearth, the extra consumption of wood during the finishing roasting could doubtless have been materially lessened; but what made this plan impracticable was the change the ore underwent in passing through the Stetefeldt furnace, which interfered with attempts to obtain a high chlorination result afterwards.

The re-roasted ore was of a red-brown colour, smelled of chlorine and did not emit any sulphurous acid gas, but it still consisted principally of the little globules, of which mention has been made, quite a large number of which remained black, no matter how long the ore was kept in the furnace. Some were magnetic, but the majority were not. Between the fingers, the re-roasted pulp felt sharp, like powdered glass. The temperature was kept at a proper degree, and there was an abundant draught, the dust-chambers and furnace having been previously cleared. Still it was impossible to obtain more than 44·2 per cent. of chlorination, owing undoubtedly to the silicates formed during the roasting in the Stetefeldt furnace. A jet of steam* introduced into the reverberatory hearth attached to the modified Howell furnace considerably improved the results, but still did not give sufficient satisfaction.

* The introduction of steam through the bridge of the furnace generally increases the fuel consumption.

The average of thirteen charges showed:—average of re-roasted ore, 31·0 ounces per ton; average of leach-tailings, 10·37 ounces per ton; average of chlorination, 66·6 per cent. The ore still contained a considerable number of globules, but they had changed their colour to red-brown, and between the fingers the ore felt soft and not so sharp and glassy as when roasted without steam. The Howell furnace alluded to above was not a regular Howell, but consisted of a revolving cylinder of uniform diameter with a shell of boiler-plate lined with bricks throughout its length (the principle on which it works is identical, however, with the Howell furnace), and had a reverberatory hearth in front, attached at the end.

The Howell, like the Stetefeldt furnace, requires the salt to be mixed with the ore before entering the furnace. With some ores this is immaterial, but it is a matter of the greatest importance for the del Oro ore. If the salt is previously added the ore becomes sticky, encrusts the furnace rapidly, and leaves the furnace mostly in lumps imperfectly chloridized. If the ore is charged without salt it remains dry and sandy, but a very annoying separation takes place. The fine particles are carried by the draught into the dust chambers, and only the coarse sand passes through the furnace without being sufficiently desulphidized. If, then, salt be added in the drop-pit, only a small percentage of silver becomes chloridized. The best results gave only 29 per cent. of chlorination. In order to diminish the separation, 2 per cent. of salt was added to the ore in the battery. This small percentage of salt made the ore sticky enough to diminish considerably the dusting, without causing the formation of lumps or too heavy encrustations.

By this method, the chlorination improved considerably, the average of three days' run being 67 per cent., but Mr. Hofmann convinced himself notwithstanding, that the Howell furnace, as such, could not roast the del Oro ore. The results were not sufficiently uniform and reliable; and though the roasted ore was left in a good condition, the average could not be brought above 67 per cent. An alteration being apparently required, which would give the ore more roasting time and allow of a better regulation of the temperature, Mr. Hofmann made the following changes. In front of the furnace, he constructed a shallow drop-pit and a fireplace, and on one side of the drop-pit, communicating with it, he built a small reverberatory hearth, 6 feet by 8 feet, the bottom of both being on the same level.

The reverberatory hearth contained one working-door and a 24 inches fireplace. When enough ore had accumulated in the pit to make a charge for the reverberatory hearth, it was pushed in with a hoe, each charge consisting of about 1,400 lbs. When starting the furnace, a strong fire

was kept up in both fireplaces, but after the process was in operation the fire in front of the cylinder was much lowered; in fact so much so, that half the grate bars remained bare of wood. Only now and then a thin stick of wood was added, just enough to prevent the drop-pit getting chilled. Two per cent. of salt was added to the ore in the battery.

If the roasting be properly conducted, the blue flame of the ignited pyrites can be observed in the back part of the cylinder. Next to it, and reaching beyond the middle of the cylinder, the ore assumes a higher temperature, forming a belt of bright red heat. The part of the furnace next the fire (nearly one-third of the whole length) should look dark. The furnace is mostly heated by the combustion of the sulphides, and receives but little supply from the fireplace and reverberatory hearth; in fact the ore in the cylinder should be left as much as possible to roast in its own heat. This is a very important condition to maintain, the object being to convert as much as possible of the galena and zinc-blende into sulphates and oxides before generating chlorine, and to avoid until then as much as possible the decomposition of the iron-salts. This can only be done by maintaining a low heat after the combustion of the pyrites.

An excess of heat is invariably connected with an excessive loss of silver by volatilization and by a low chlorination. Galena and zinc-blende roast quicker and better in a low than a high temperature. When the ore leaves the cylinder and drops into the pit, it should be of a dull red heat, while the colour after cooling should be dark yellow-brown. If the temperature be so kept, neither the odour of chlorine nor much of sulphurous acid can be detected. At an increased heat, sulphurous acid is again evolved strongly, showing that the oxidation is not yet completed. As the temperature in the cylinder is mostly produced by the combustion of the sulphides, the chief means of regulating the same is the feed. If too much ore enters the furnace, the belt of bright red heat increases, advancing more and more towards the front, and finally the whole furnace assumes this temperature. The ore dropping into the pit is very hot, emits heavy fumes and overheats the pit. If it be then removed into the reverberatory hearth, it takes a very long time to finish, necessitating an interruption in the feed of the cylinder.

On the other hand, if insufficient ore be charged, the belt of bright red heat gets smaller and moves towards the back end of the furnace. When the properly-prepared ore enters the reverberatory hearth, the salt is added and the temperature increased. It commences to fume, and swells without forming more lumps than an ordinary ore. In the beginning, strong fumes of sulphurous acid are emitted, but soon cease and chlorine appears.

The charge is finished if the fumes assume a mild and sweetish smell of chlorine; as long as they smell strong, roasting must be continued.

Mr. Hofmann made a series of experiments to ascertain the smallest amount of salt practicable, and found that 4, 6, 8, and 10 per cent. gave about equal results; 12 per cent. commenced to make the ore too sticky and produces less chlorination; 3 per cent. was sufficient if the roasting were very carefully conducted, but then only 1 per cent. had to be added in the battery and 2 per cent. in the furnace; 4 per cent., however, was safer, for then the result did not depend so much on the skill and goodwill of the labourers. The roasting capacity of the furnace proved much less for the del Oro than for ordinary ore. The cylinder was only 24 feet long, which could not roast more than 8½ tons per 24 hours; but even with a 32 feet cylinder, Mr. Hofmann states he did not expect to roast more than 12 tons per day.

Each charge had to remain 2 hours on the reverberatory hearth. Though the ore was roasted in the reverberatory hearth at a somewhat increased heat, it could not be raised beyond dull red without losing too much silver by volatilization.

If the silver-bearing minerals of an ore be not of great density and decrepitate in the heat, the ore can be crushed coarse without endangering the subsequent roasting, but if the principal silver-bearing mineral, like the zinc-blende in the del Oro ore, be of great density and does not decrepitate, it is of the greatest importance to crush fine.

A series of experiments was made by Mr. Hofmann with ore crushed through No. 20 and No. 40 screens. He found that the ore crushed through No. 20, required a much longer time and was 27 per cent. less chloridized than the ore crushed through No. 40 screen. The material which passes through a battery-screen of a certain number, is much finer than the size of the meshes. Heavy ore makes a much finer pulp through the same screen than lighter ore. The pulp of the del Oro ore obtained by crushing through battery-screens Nos. 20 and 40, when sieved through sieves of different fineness, showed the following results:—

Battery Pulp when Sifted through Sieve. No.	Crushed through Battery Screen No. 20. Percentage of Material passing through the Sieve.	Crushed through Battery Screen No. 40. Percentage of Material passing through the Sieve.	Crushed through Battery Screen No. 20. Percentage of Material remaining on the Sieve.	Crushed through Battery Screen No. 40. Percentage of Materia remaining on the Sieve.
30	93·8	100	6·2	0·0
40	87·3	100	12·7	0·0
60	78·8	98·95	21·2	1·05
80	71·2	93·80	28·7	6·20
90	67·1	90·50	32·9	9·50

These figures show how exceedingly fine a heavy ore is crushed (dry) in a battery, even through a screen with comparatively coarse meshes. Though 67·1 per cent. of the material crushed through screen No. 20 was finer than sieve No. 90, the average chlorination of quite a number of comparative roastings was 27 per cent. less than that of ore crushed through a No. 40 screen. This indicates how essential it is to crush such ores fine.

Mr. Hofmann found that in order to chloridize the del Oro ore well, it had to be crushed through a No. 40 battery-screen, which furnished a pulp of which only 9½ per cent. was coarser than sieve No. 90. To produce such a fine pulp with rolls, would require such exceedingly fine screens that their use would be here impracticable. The crushing capacity of the battery was not much diminished by using No. 40 screen instead of No. 20, principally because the ore was so heavy.

For comparison, Mr. Hofmann ran 5 stamps, 12 hours, crushing through No. 20, and 5 stamps through No. 40, and found :—

No.	Lbs. of Pulp.
20 with salt furnished...	8,100
40 with 4 per cent. salt furnished ...	7,488
Difference in favour of No. 20 screen	612 lbs., or 7½ per cent.

A very small reduction, if the great advantage gained for roasting is considered. Some ores gain much in chlorination of the silver, if left hot in a pile for several hours; this is mostly the case when an ore is insufficiently roasted, or when the nature of an ore is such as to require long roasting at low heat.

Additional chlorination can be produced by moistening the ore and leaving it for several hours in the pile; this is usually the case if the ore contains copper. Roasted ore containing caustic lime should not be left moist on the cooling-floor, hence the del Oro ore could not be wetted on the cooling-floor without decomposing some of the silver chloride. Mr. Hofmann remarks that the most important additional chlorination takes place, however, during base-metal leaching.

In the Silver King ore, Arizona, this additional chlorination amounted to nearly 6 per cent. Mr. Hofmann therefore filled the vat before charging about one-third with water and dumped the ore hot into it, thus producing a hot base-metal solution, and made the observation that by it not only the decomposition of the silver chloride was avoided, but that a considerable increase in the silver chlorination took place, amounting in some instances to 12·9 per cent. If the original chlorination was 75 per cent. or more the increase was, however, much less. By adding some cupric chloride to the water in the vat before dumping in the ore, badly roasted

charges gained as much as 34 to 38 per cent. (see charges, Nos. 9, 15, and 16 in annexed table). These are very important observations, and give the operator a means of correcting badly roasted charges.

The following table is a careful record of the results obtained in roasting the San Francisco del Oro ore in the Howell furnace modified by Mr. Hofmann, representing a two weeks' run. As each tank-charge contained the whole ore of 24 hours' roasting it offered a good opportunity of following each charge through the process from the condition of raw ore to tailings, and of ascertaining for each the loss by volatilization, etc.

Taking the average of the results we find the silver chlorination when the ore left the furnace was 68·4 per cent.; the additional chlorination was 13·3 per cent.; or a total of 81·7 per cent. The low average chlorination of the ore when leaving the furnace, was caused by the three badly-roasted charges 9, 15, and 16. The other eleven charges gave an average of about 75 per cent.

Mr. Hofmann remarks—"The total or actual chlorination of 81·7 per cent. may seem to be low, but if we consider that the ore is of low grade, averaging only 28·85 ounces per ton, and that 1 per cent. represents only 0·28 ounce of silver, and also consider that the ore contains about 37 per cent. of zinc-blende, and 13 to 19½ per cent. of galena, which carry all the silver, and that the ore was pronounced as being too refractory for chloridizing-roasting, we have to count the work done by the modified Howell furnace, as very satisfactory."

In order to ascertain the loss of silver by volatilization we have first to ascertain the loss in weight which the ore sustains during roasting. This can be done by the muffle test described by Mr. Hofmann in *The Engineering and Mining Journal*, New York, of April 23rd, 1887.* "Ten grammes of the raw pulp containing salt are placed in a roasting dish and roasted in the muffle for half an hour or an hour, then the sample is removed from the muffle, allowed to cool, weighed, placed back in the muffle, and roasted again for half an hour, then weighed again. This is repeated till two weighings are alike, or till in the last half hour of roasting, the ore does not lose more than 2 or 3 milligrames. The difference between the original weight and that of the last weight expressed in percentage gives the highest possible loss which the raw ore can sustain."

Ten grammes of a sample of roasted ore corresponding with the sample of raw pulp are placed in a roasting dish, and also roasted in the muffle until two weighings agree or differ after half an hour's additional roasting not more than 2 or 3 milligrammes. The difference between the first weighing and the last (expressed in percentage) gives the weight which the

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WORKING RESULTS OBTAINED IN ROASTING.—EXPERIMENTS WITH THE MODIFIED HOWELL FURNACE.

Date, 1888.	Per Cent. of Salt used.	Value of Raw Ore.	Value of Raw Ore containing Salt.	No. of Tank-charge.	Value of Roasted Ore.	Value after Leaching in the Laboratory.	Per Cent. of Chlorination, when the Ore left the Furnace.	Value of the Vat-tailings.	Per Cent. of Actual Chlorination, Calculated from the Vat-tailings.	Loss in Weight which the Ore Sustained during Roasting.	Loss of Silver by Volatilization.	Loss of Silver by Tailings.	Per Cent. of Chlorination Gained in the Vat.	Number of Ozs. of Silver Extracted.	Per Cent. of Actual Silver Extraction.	Remarks.
		Ozs. per Ton.	Ozs. per Ton.		Ozs. per Ton.	Ozs. per Ton.	Per Cent.	Ozs per Ton.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Ton.	Per Cent.	
April 20	5½	34·21	32·33	9	31·34	11·66	37·2	7·75	75·3	2½	5·2	24·5	38·1	24·04	70·3*	Assorted ore.
„ 21	6	34·50	32·43	10	30·91	9·04	71·8	7·26	76·6	2½	7·1	21·8	4·8	24·52	71·1	
„ 23	5	35·09	33·34	11	31·49	8·89	71·8	4·33	84·7	2½	7·8	14·1	12·9	27·40	78·1	
„ 24	5	33·76	32·07	12	29·74	8·16	72·6	3·29	89·0	2½	9·5	10·0	16·4	27·17	80·5	
„ 25	5	31·67	30·09	13	29·59	6·99	76·4	3·57	88·0	2½	4·1	11·5	11·6	26·72	84·4	Ore slightly assorted.
„ 26	5	29·09	28·64	14	29·16	7·58	74·0	4·25	85·5	3½	1·7	14·3	11·5	24·55	84·4	
„ 27	4½	27·35	26·02	15	25·95	13·27	51·9	4·25	84·7	3½	3·7	15·7	32·8	22·04	80·6*	
„ 28	4	28·32	27·19	16	24·49	12·89	47·4	4·48	81·8	3½	13·0	15·8	34·4	20·16	71·2*	
„ 29	4	26·02	24·98	17	22·16	4·95	77·7	4·76	78·7	3½	14·2	19·0	1·0	17·38	66·8	
„ 30	5	25·43	24·42	18	22·74	5·24	77·0	4·48	80·3	3½	10·1	18·3	3·3	18·20	71·6	Unassorted ore, just as it was broken-down in the mine.
May 1	5	24·37	23·15	19	21·57	4·66	78·4	4·84	77·6	3½	10·0	20·0	0·8	17·05	70·0	
„ 2	5	23·94	21·97	20	22·45	5·39	76·0	3·87	82·8	3½	1·3	17·0	6·8	19·46	81·3	
„ 3	4	24·28	23·31	21	22·01	5·83	73·6	4·84	78·1	3½	8·7	20·0	4·5	17·31	71·3	
„ 4	4	26·02	24·98	22	21·87	6·12	72·1	4·30	80·4	—	15·5	16·6	8·3	17·66	67·9	
	4·7	28·85	27·44		26·10	7·90	68·4	4·76	81·6	3	7·9	17·0	13·3	21·69	74·9	Averages.

Charges marked * were treated during base-metal leaching with cupric chloride.

roasted ore is still capable of losing if subjected to prolonged roasting. If we deduct this last loss from the highest possible loss, we obtain in percentage the loss in weight by volatilization which the ore has suffered during roasting in the furnace.

Owing to a great part of the lead and zinc sulphides being converted into sulphate, the San Francisco del Oro ores lose but a small percentage of their weight during roasting. The tests showed a loss of $2\frac{1}{2}$ to $3\frac{1}{2}$ per cent. With these figures and the assay value of the raw and roasted ore, the loss of silver by volatilization was calculated. The extremes were 1·7 and 15·5 per cent., and the table illustrates how variable this loss is. Mr. Hofmann says he found the del Oro ore more sensitively disposed for such losses than many others, even antimonial ores. The least increase of temperature above dull red causes a heavy loss, even if this increase of temperature lasts only for a very short time. Two or three thin sticks of wood thrown on the fire before they are needed may materially increase the loss. The loss by volatilization is not in direct proportion to the percentage of chloridized silver. In proof of this Mr. Hofmann instances the results of roasting Las Yedras ore at high and low temperatures in Brückner furnaces :—

High temperature, averages of 31 consecutive working-days, January 13th to February 12th, 1887.

	Furnace No. 3. Per Cent.	Furnace No. 4. Per Cent.	Average of both Furnaces. Per Cent.
Chlorination	71·3	74·2	72·7
Loss by volatilization ...	18·3	17·6	17·9

Low temperature, averages of 30 consecutive days, June 1st to July 1st, 1887.

	Furnace No. 3. Per Cent.	Furnace No. 4. Per Cent.	Average of both Furnaces. Per Cent.
Chlorination	81·7	81·3	81·5
Loss by volatilization ...	1·7	0·7	1·2

By comparing these average results we find an increase of	Per Cent.
chlorination, in favour of low heat, of	8·8
A decrease of loss by volatilization caused by low heat ...	16·7
Total increase of production	25·5

The roasted ore contained only a small percentage of lumps, not hard, but porous, which fall to powder if kept in contact with water for some time. If the ore be left dry in a pile, it hardens. If left undisturbed for a week or two, it becomes so hard that a pick is necessary to loosen it. In water, however, it softens easily again. The colour is usually red-brown,

but occasionally, if the ore contain less iron pyrites, it is yellow-brown. The heavy metals in the Yedras ore are principally converted into sulphates, and only a small proportion are present as chlorides.

The following is an analysis of the unassorted ore after roasting with 5 per cent. of salt :—Gold, traces ; silver, 0·09 per cent. ; lead, 9·00 per cent. ; iron, 6·00 per cent. ; zinc, 22·45 per cent. ; caustic lime, 5·65 per cent. ; antimony, 0·75 per cent. ; copper, 0·60 per cent. ; cadmium, 0·10 per cent. ; alumina, 3·09 per cent. ; caustic soda, 3·79 per cent. ; sulphuric acid, 13·16 per cent. ; chlorine, 0·88 per cent. ; soluble silica, 8·00 per cent. ; and insoluble gangue, 18·61 per cent.

Mr. Hofmann mentions an easy method of removing crusts from continuous discharging-furnaces, which can be done while the furnace is in operation. In the masonry at the back an opening is made, so located that with a long iron spade the interior of the cylinder can be reached just below the feed-pipe. Through this opening, 1½ dozen firebricks are introduced while the cylinder is revolving, and in moving slowly forward shove off the crust clean down to the lining. The crust when hot is soft and yields to the weight of the moving bricks. It is not necessary to interrupt the feeding. If bricks should be found too light, heavier, but not too large, pieces of castings may answer.

Owing to the large quantity of sulphides in the ore and the very low temperature at which the del Oro has to be roasted, the consumption of wood is very small. After the furnace is heated and the cylinder encrusted, it takes hardly any fire in front of the cylinder to maintain the proper temperature. In the reverberatory a little more fire is needed, but much less than with ordinary ores. Mr. Hofmann states that he weighed the wood during a couple of weeks, and found the total consumption during this period to be 220 cargas (at 300 lbs.). With this quantity of wood he roasted 115·8 tons, and therefore used 1·8 cargas per ton. 12 cargas of Parral wood being equal to 1 cord; if we express the consumption in cords, we find that with 1 cord of wood the furnace roasted 6·3 tons of ore. The cost of roasting in the modified Howell furnace (roasting 8½ tons per 24 hours) is as follows :—

	Dollars.
2 head roasters at $1·50	3·00
4 helpers at 90 cents	3·60
4 per cent. salt, 680 lbs. at 1·27 cents ...	8·63
15·7 cargas wood at 75 cents	11·77
Steam-power, 10 cargas of wood at 75 cents	7·50
Oil, light, tools, etc.	2·00
Management, office, mechanic, assay office	1·78
Total	38·28

And the cost per ton, $4·50, Mexican currency.

To ascertain the cost of steam-power a separate boiler was used for the furnace. It is evident that by using a boiler for only one small furnace, the expense per ton of ore will be much greater than if with the same boiler and engine several large furnaces are operated, but this was the only way of getting an estimate.

The steam for working the pump and preparing the calcium sulphide were supplied by the same boiler and had to be charged to roasting. It is included in the above statement, and will therefore not appear in the statement of cost of lixiviation.

As the statement of cost is made for only 8½ tons per day, it would be misleading if the whole expense for management, mechanics, assay office, etc., were charged to the 8½ tons, as these expenses would remain the same for 100 tons a day—the intended capacity of the new mill. The proportion chargeable on 8½ tons, treating 100 tons per day, has therefore been debited to this item, but as there are three departments in the mill, viz., stamping, roasting, and leaching, each department has to be charged with one-third of this expense; 1·78 dollars represent therefore one-third of the cost.

The figures contained in the preceding table are the results of experiments obtained with different treatment as regards salt, temperature, time, etc., and the averages do not therefore represent the best obtainable results. This is especially the case in regard to loss by volatilization, which should be reduced as the men gain experience in handling the furnaces. They were good enough, however, to secure a profitable reduction of the ore, and may be accepted as a basis for calculations and estimates.

In order to increase the roasting capacity to the stamping power of the mill and for the sake of further experiments, Mr. Hofmann erected four double-hearth reverberatory furnaces (the lower hearth of 220 square feet surface, and the upper of 210 square feet surface) to supplement the modified Howell furnace.

Each double furnace took four charges of 1 ton each; when one charge was finished, all the others were moved forward, and on the first hearth a new charge dropped through the opening in the arch. From the second hearth the charge was dropped on the lower hearth, through an opening in the bottom near the working-door. The upper hearth was exclusively used for oxidizing-roasting, and 4 per cent. of salt was added while the charge was dropped on to the lower hearth. Every 2½ to 3 hours a charge was done, and therefore each double furnace roasted from 8 to 10 tons per day, according to the quantity of lead the ore contained. The chlorination results were so near those obtained in the modified Howell

furnace that it is not necessary to give details, but Mr. Hofmann records the observations he made on a charge submitted to a prolonged oxidizing-roasting. They are too long to be given here, but will be found fully described in *The Engineering and Mining Journal*, New York, of February 23rd, 1889.*

He found that the ore during oxidizing-roasting sustained a loss of silver by volatilization of not less than 13 per cent. (not taking into calculation the loss in weight), and that this loss principally occurred during the seventh hour, at the time the ore assumed a dark red-brown colour.

Only a part of the soluble silver, he concludes, was present as sulphate, viz., 25·2 per cent. Of the 58·8 per cent. which was soluble, 33·6 per cent. was some other silver salt, not soluble in water, but soluble in sodium hyposulphite, probably silver antimoniate.

The reverberatory furnaces were built in pairs, two being connected with one main flue.

During four weeks the wood consumed by one pair of these furnaces was weighed. In this time 507 tons of ore were roasted at a consumption of 672 cargas of wood, or 1·3 cargas per ton. If expressed in cords, we find that one cord of wood roasted 9 tons of ore, an extremely small consumption.

The cost of roasting in the reverberatory furnace, for two double-hearth furnaces roasting 18 tons per day, was as follows :—

	Dollars.
2 head roasters at $1·50	3·00
16 roasters at $1·00	16·00
2 wood carriers and ore wheelers at 90 cents	1·80
2 carmen for raw ore chargers at 90 cents	1·80
23·4 cargas of wood at 75 cents	17·55
4 per cent. salt, 1,440 lbs. at $1·27 ...	18·28
Tools, etc.	4·00
Management, office, mechanics, assays ...	3·77
Total	66·20

And the cost per ton, 67 cents, Mexican currency.

To form an opinion as to which roasting-furnace is the most suitable for an ore like the San Francisco del Oro, we have to take into consideration only the modified Howell and the reverberatory furnaces. The Stetefeldt furnace did not answer, and the Brückner furnace was not tried, as experience has shown that a large Brückner furnace could not roast more than 5 or 6 tons of such a heavy ore. Both the modified

* Page 190.

Howell and the reverberatory furnace gave about the same results, and the loss by volatilization was also nearly the same. The cost, however, was different, viz., 4·50 dollars per ton in the former, and 3·67 dollars in the latter, being a difference in favour of the reverberatory furnace of 83 cents per ton.

This ought to decide; still there are, besides the merits of the furnace, other circumstances to be taken into consideration. The reverberatory furnace requires quite a number of skilled hands, and the result depends much more on the skill and goodwill of the men, than with the modified Howell. Mexicans are, as a rule, if properly handled, very good workmen, notwithstanding that they receive such moderate wages, but they like to lay off on Sundays and feast days. On such days quite a number of inexperienced substitutes will be found working at the furnaces, and the roastings consequently suffer. The main trouble, however, occurs every year in spring and autumn, the time of planting and harvesting corn, when many leave the camp, and the mill is left short of hands. For this reason it may be more advisable in a large mill to use the modified Howell furnace, notwithstanding that the reverberatory furnace works cheaper and creates much less flue-dust. The Howell furnace forms a great deal of flue-dust, which is far from being roasted even if the furnace be provided with an auxiliary fireplace, as in the Howell-White furnace.

Lixiviation.

Base-metal Leaching.—The roasted ore is moistened with water and charged into wooden vats, in charges of 10 to 15 tons, though much larger charges are sometimes worked. The vats are best furnished with a central discharge, around which a filter-bottom is arranged in the shape of a very flat funnel. The filter-cloth is kept in place by ropes driven into grooves around the discharge-hole and inner periphery of the tub, near the filter-bottom. The vat is provided with an outlet under the filter, and has a slight inclination towards this point.

The charge of roasted ore is leached with water to remove the soluble base-metal salts. Water does not dissolve silver chloride, but a concentrated solution of base-metal chlorides does, and it is therefore advisable not to make the leaching-vats too deep, as otherwise a too concentrated base-metal solution is produced by the water in descending through a thick layer of ore. The base-metal leaching is completed when a few drops of calcium polysulphide poured into some of the outflowing solution does not produce a precipitate. This part of the process, according to the character of the ore, takes 4, 8, or 10 hours.

With the San Francisco del Oro ore, which filters well, the rate of filtration during base-metal leaching was 1 inch in 6 minutes or 10 inches per hour. With a vat 10 feet 2 inches in diameter, representing a filtering surface of 81·01 square feet, and a thickness of charge of 2·41 feet, the above rate was equivalent to 67·24 cubic feet, or 504·3 gallons per hour; leaching time, 8 hours = 4,034·4 gallons; 2 feet of water in the vat into which the ore is dumped = 1,215·2 gallons extra. Second washing, rate of filtration, 8⅜ inches per hour; time, 1½ hours = 647·1 gallons. Total for one charge of 8·39 tons, 5,896·7 gallons, or 703 gallons for each ton of ore. It has been mentioned that, on account of the roasted ore containing considerable caustic lime, it was not moistened as usual on the cooling-floor but dumped dry and hot into about 2 feet of water in the vat. Besides other advantages, this way of charging shortens the base-metal leaching time by nearly an hour, without reducing the rate of filtration. When the charge is in, the water ought to cover the ore about 1 inch. Leaching is then commenced from below,* and the solution diluted above. As soon as the solution is allowed to flow out below, a stream of clear water has to be turned on above and continued for about an hour, in order to produce a greater dilution. Then the influx is interrupted until the water sinks to the level of the ore, when water is again allowed to flow in. The base-metal solution of the del Oro ore shows an acid reaction. The leaching with water is continued until a few drops of calcium sulphide added to the solution produces only faint white clouds. These white clouds continue to show for hours, and are caused by the reaction of calcium sulphide and sodium sulphate precipitating gypsum. It takes a long time to leach alkaline salts contained in porous substances, thus it happens that while all the soluble metal salts have been dissolved and removed, the outflowing solution still contains and continues to contain sodium sulphate.

From a chemical standpoint, it would be advisable to continue leaching with water till all the sodium sulphate is removed, but in practice this would delay the process too much, and the silver leaching is therefore commenced as soon as all the heavy metal salts are removed. The sodium sulphate which still remains in the ore after leaching with water, enters the stock solution during the subsequent silver leaching. This has not a very injurious effect if calcium sulphide is used as a precipitant, because sodium sulphate is decomposed and gypsum precipitated.

The stock solution is therefore freed from sodium sulphate after every

* This is done to precipitate on to, and through the ore any chloride of silver that might be dissolved by a too concentrated solution of the base-metal chlorides.

precipitation, and the only resulting disadvantage will be that the precipitate will contain more gypsum. But if sodium sulphide is used as a precipitant, the effect of the sodium sulphate is very injurious, as it remains and accumulates in the stock solution and soon reduces its dissolving energy for silver chloride, rendering a prolonged leaching with water necessary. By inserting, in the outlet of the vat, a small rubber tube (provided at the end with a glass tube drawn to a fine point), and by leaving this tube in the outlet during the whole time of base-metal leaching, Mr. Hofmann obtained from the outflowing solution a very fine stream which, collected in a proper vessel, gave a true sample of the solution of about 3 or 4 gallons. This sample contained in 1,000 cubic centimetres:—

	Grammes.		Grammes.
Silver	0·0036	Lead	—
Cadmium ...	0·806	Sulphuric acid ...	18·708
Zinc	2·105	Chlorine	7·173
Iron	0·008	Lime	0·754
Copper... ...	Trace		

Silver Leaching.—The base-metal salts being removed, a stream of diluted solution of sodium hyposulphite is allowed to enter on top of the ore, which readily dissolves the silver chloride. When the outflowing solution shows indications of silver, the stream is conveyed to special precipitating-tanks, in which the silver is precipitated as silver sulphide by an addition of calcium polysulphide or some other reagent.

To ascertain the exact time when the hyposulphite solution, which follows the water, commences to appear at the outlet, a matter of importance, it is ordinarily tested with calcium polysulphide, and some operators indulge in the bad habit of testing it with the tongue, and are guided by the sweetish taste which the solution has if it contains silver. The following test is much superior, and is very sensitive and convenient:—A small strip of starch paper is dipped into a solution of iodine and then held in the outflowing stream. If the blue colour disappears it is a sign that the liquid contains a hyposulphite salt, and the stream must be turned into the precipitation-vats. The test is only applicable, however, when the base-metals are leached with cold water, as hot water discolours the blue paper. To facilitate and hasten the settling of the silver precipitates, the precipitation-tanks are provided with machine stirrers, by which the solution can be vigorously agitated.

After precipitation, the sodium hyposulphite solution is decanted after the precipitate has settled in tanks placed at a lower level. From these tanks the clear solution is pumped up to storage-tanks and

is ready to be used again. When all the soluble silver is extracted, the solution of the hyposulphite is allowed to run out till it disappears under the surface of the ore, when clear water is introduced again to displace all the hyposulphite solution; after this second leaching with water, the tailings are sluiced out through the central discharge, and the tank is ready for another charge of ore. The time for silver leaching varies according to the character of the ore from 8 hours to 2 or 3 days.

In the ordinary lixiviation process, calcium sulphide is the ordinary and best precipitant. This salt, however, cannot be used for that purpose in the Russel process, and sodium sulphide therefore takes its place. In the Kiss process the ore is leached with calcium hyposulphite, and the silver is precipitated with calcium sulphide. Leaching with sodium hyposulphite and precipitating with calcium sulphide (the ordinary method) is really a combination of the Patera and Kiss processes. When enough silver precipitate has accumulated on the bottom of the precipitating-tanks, it is drawn off and generally strained through a filter-press. The black silver cakes are then taken out, dried in a warm room or drying-oven, and introduced into a muffle or calcining-furnace to burn off the sulphur. After the blue flame has disappeared, heating must continue for several hours at a dark red heat.

The roasted cakes may be melted with lead in a cupelling-furnace, and refined or melted in graphite crucibles, in lots up to 300 lbs. in weight. What sulphur remains must be removed by melting it off with metallic iron introduced into the pot, an iron matte being formed, which rises to the surface and is skimmed off. The surface of the silver is then cleaned by adding some bone ash and borax, or borax alone, which is also skimmed off, and the silver dipped out or poured into moulds.

The quantity of hyposulphite solution required for treating the San Francisco del Oro ore is as follows :—The diameter of the vat is the same as for base-metal leaching (previously described), viz., 10 feet 2 inches. Rate of filtration, $8\frac{3}{8}$ inches per hour or 57·52 cubic feet or 431·4 gallons per hour. Time of silver leaching, 4 days or 96 hours or 41·414 gallons per charge of 8·39 tons, or 4,935 gallons for each ton of ore, 658 cubic feet.

At the Cusi mill, Mr. Hofmann states the vats are 12 feet in diameter, taking a charge of about 8 tons of ore. At a filtering rate of 8 inches per hour the volume of outflowing solution amounts to 74·6 cubic feet per hour, or 3,058·6 cubic feet in 41 hours for a charge of 8 tons, or 382·3 cubic feet or 2,867 gallons per ton of ore, employing the Russel process, taking the figures given by Mr. Stetefeldt. Mr. Hofmann, however, states

that he found the average leaching time at the Cusi mill to be 53 hours, which at the filtering rate of 8 inches per hour gives 494·2 cubic feet or 3,706 gallons per ton of ore.

The Stock Solution.

Mr. Hofmann appears to have been the first to introduce the practice of using sodium hyposulphite as a solvent and calcium sulphide as a precipitant. Calcium sulphide contains a considerable amount of (about 6 per cent.) calcium hyposulphite, even if freshly prepared, which in precipitating is introduced into the stock solution. It was therefore supposed that by using calcium sulphide as a precipitant the stock solution would be gradually converted into a solution of calcium hyposulphite, but Mr. Hofmann denies that this is the case, and states that the stock solution remains as sodium hyposulphite. He explains it in this way: At the time when base-metal leaching is usually interrupted, the ore still contains sodium sulphate. Calcium hyposulphite and sodium sulphate form sodium hyposulphite and calcium sulphate, which precipitates

$$CaS_2O_3 + Na_2SO_4 = CaSO_4 + Na_2S_2O_3.$$

If therefore the stock solution after precipitation, containing calcium hyposulphite, is rinsed, and comes in contact with the sodium sulphate contained in the roasted ore, the outflowing solution will have its calcium hyposulphite substituted by sodium hyposulphite, leaving the precipitated gypsum in the ore.

Thus the stock solution, even after continued use, still consists of sodium hyposulphite. It takes longer boiling to manufacture calcium sulphide than to prepare sodium sulphide, but 3½ to 4 hours' boiling is sufficient, and the consumption of steam is small to maintain the solution, once it is boiling, at that temperature. The calcium sulphide appears to have the great advantage that it frees the stock solution from the very injurious presence of sodium sulphate, for if calcium sulphide is brought in contact with sodium sulphate, sodium sulphide is formed, which goes into solution, and the calcium sulphate is precipitated; thus all the sodium sulphate which the stock solution receives from the ore during lixiviation is decomposed, while sodium sulphide if used as a precipitant not only leaves the sodium sulphate undecomposed, but even furnishes an additional supply of this salt, and assists in charging the stock solution with it, diminishing its dissolving energy for silver chloride, and causing quite a large consumption of sodium hyposulphite.

If in preparing calcium sulphide, boiling is not continued beyond 3 or 4 hours and precipitation is properly executed, the original stock

solution can be maintained effective for several years without adding any fresh sodium hyposulphite to it. It will be even found that the solution increases in strength, and in order to keep it at standard strength it requires to be diluted from time to time with water. This cannot be accomplished if sodium sulphide be used. The loss of hyposulphite by decomposition and other causes is in fact more than replaced by the supply derived from the precipitant.

Mr. Hofmann gives a table* showing the tendency of the sodium hyposulphite solution to increase in strength, as above explained, and remarks that it is quite an important financial item whether its standard strength (0·50 per cent.) is kept up by adding water or sodium hyposulphite. At the Cusi mill, he used 765 lbs. of sodium hyposulphite for the preparation of the stock solution, working with it 2,011 tons of ore, and left the solution in perfectly good condition, only a little stronger than when originally prepared, without adding any extra sodium hyposulphite, while Mr. Daggett, who used sodium sulphide, reports a consumption in the Cusi mill of 3 to 7 lbs. of sodium hyposulphite per ton of ore.

Mr. Hofmann states that in working 2,011 tons of Cusi ore containing 45·2 ounces of silver per ton, the cost of calcium sulphide per ton of ore was as follows :—

	Cents.
Sulphur, 3·92 lbs. at 7 cents	27·4
Lime, 8·25 lbs. at 1 cent	8·2
Total	35·6

or 28·7 cents less than the cost of sodium sulphide (reckoned at 64·3 cents) treating an ore containing 35·1 ounces of silver. Reducing the above figures to what they would be in working an ore containing 35·1 ounces of silver per ton, they would stand :—

	Cents.
Sulphur, 3·02 lbs. at 7 cents	21·1
Lime, 6·40 lbs. at 1 cent	6·4
Total	27·5

or 36·8 cents less than the cost of sodium sulphide, which is equal to a saving of 57·2 per cent. Taking into consideration the fact that the use of sodium sulphide causes also a consumption of 3 to 7 lbs. of sodium hyposulphite per ton of ore, it is apparent that sodium sulphide is more expensive, and that calcium sulphide is preferable as a precipitant, regardless of other advantages, for the important reason that it keeps the stock solution free from sodium sulphate.

* *The Engineering and Mining Journal*, New York, vol. xlvii., page 236.

The calcium polysulphide is manufactured at the works, by boiling 2 parts of the fresh lime with 1 part of pulverized sulphur in water for 3 to 4 hours, in deep tanks made of boiler iron, into which steam is directly introduced. The consumption of sulphur is from 2 to 7 lbs. per ton according to the ore.

The Precipitate.

Mr. Hofmann gives the analysis of two different lots of roasted precipitate of del Oro ore:—

	Per Cent.	Per Cent.		Per Cent.	Per Cent.
Gold ...	0·04	0·014	Zinc ...	4·30	13·86
Silver ...	19·00	21·60	Lime ...	3·88	3·62
Lead ...	30·64	21·10	Sulph. acid	6·10	6·18
Copper	11·55	4·44	Sulphur	14·90	19·37
Cadmium	3·45	1·20	Insoluble	5·45	4·96
Iron ...	0·72	2·68			

The two lots of precipitate differ, as will be seen, considerably with regard to lead, copper, cadmium, zinc, and sulphur, due partly to variations in the character of the ore, of which the precipitate is the resulting product, and to a great extent to variations in roasting.

The quantity of sulphur depends on the length of time the precipitate is subjected to roasting in the reverberatory furnace. To avoid loss by volatilization, the precipitate was left in the furnace only until the blue sulphur flame ceased. The percentage of lime (3·88 to 3·62) is in both lots nearly the same, and shows that the value of the precipitate is not depreciated, by using calcium sulphide as a precipitant, enough to make its use objectionable. The dried precipitate contains 47·96 per cent. of sulphur. Experiments to regain the surplus sulphur by boiling the fresh precipitate with caustic soda gave very satisfactory results. They showed that 60 per cent. of the sulphur originally contained in the precipitate can be thus regained and brought into a state in which it can be directly used again as a precipitant.

Using Cupric Chloride for Badly-roasted Charges.

If an insufficiently chloridized ore be treated during base-metal leaching with a dilute solution of cupric chloride it has a very beneficial effect, as stated elsewhere, and Mr. Hofmann, in some instances, obtained a further extraction from the del Oro ore of 34 to 40 per cent. of the silver by its use. 35 lbs. of bluestone and about 70 lbs. of salt, boiled by steam for

about 15 or 20 minutes, gave a sufficient quantity of cupric chloride for a charge of 8½ tons, at a cost of 60 cents per ton of ore:—

	Dollars.
Bluestone, 35 lbs. at 12 cents ...	4·20
Salt, 70 lbs. at 1·7 cents	0·88
Total	5·08, or 0·60 cents per ton.

The cupric chloride is either added to the water contained in the tank, into which the dry and hot ore is dumped, or it is added during base-metal leaching. In the latter case it is better to apply the copper solution after the main portion of the base-metal salts have been leached out, say about 1 hour after commencing base-metal leaching, and to add it gradually in order to penetrate the whole charge. To about 6 or 8 inches of water standing above the ore, one quarter of the prepared copper solution is added, stirred, and allowed to sink through the ore. As soon as the liquid is level with the top of the ore, the outlet under the filter is closed, and again 6 or 8 inches of water is allowed to flow into the vat, to which the second quarter of the copper solution is added; this is repeated a third and fourth time, and the charge is washed in the usual way.

The first method is quicker and less troublesome, but in this case leaching from below is advisable. The solution from the del Oro ore treated in this way left the tank colourless with only a slight reaction for copper, showing that the cupric chloride was decomposed in passing through the ore. Mr. Hofmann believes that if the roasted ore of the Yedras mine were treated in this way, very good results would be obtained, without reducing the fineness of the precipitate as much as is done at present by the use of extra solution.

Cost of Lixiviation.

The works at Parral have two leaching-plants, one in connexion with the modified Howell furnace and the other with the reverberatory furnaces. Separate accounts were kept of each. The following figures refer to the Howell plant. For the consumption of sulphur no separate account could be kept, and the quantity used per ton is therefore calculated on the total amount consumed and the total number of tons of ore leached in both mills. It gives a consumption of 7 lbs. per ton of ore. One man was employed roasting the precipitate obtained from both plants, and one-half of his wages is charged in the statement. A reduction of cost, it is stated, could be effected if the precipitate were boiled with caustic soda, instead of roasting it. The item of management, office, etc., represents one-third

of the cost for 100 tons per day, calculated proportionately on 8½ tons, as explained in the cost of roasting :—

	Dollars.
Labour for charging and discharging	1·00
Two leachers at 1·00 dollar	2·00
One man preparing calcium sulphide	0·75
Sulphur, 59½ lbs. at 6 cents	3·57
Lime, 180 lbs. at 0·5 cent	0·90
One man roasting precipitate at $1·00 (one-half)	0·50
Wood for roasting precipitate	0·50
Management, office, mechanics, assay office	1·78
Oil, light, filter cloth, shovels	1·50
Steam for pump, and sulphide solution were charged to roasting	—
Total cost	12·50

or cost of leaching per ton of ore, 1·47 dollars, Mexican currency.*

Total cost of reduction, not including stamping :—

	Dollars.
Roasting in the reverberatory furnace ...	3·67
Lixiviation	1·47
Mexican dollars	5·14 per ton of ore.

Mr. Hofmann noticed that heavy brown fumes were emitted from the ore in roasting it in the muffle furnace under certain conditions, and was led to suspect the presence of cadmium in the del Oro ore in considerable quantities. Further investigations proved this to be the case. The cadmium leaches out along with the zinc, and as long as there is zinc in the solution, cadmium will be found. The fact that the cadmium is brought into solution by the regular operation of the process for extracting the silver, permits of its extraction (as a bye-product) at very small cost.

The analysis showed that the base-metal solution of the del Oro ore is remarkably free from metals which are precipitable by zinc ; if, therefore, the more concentrated part of the base-solution be conveyed into tanks (like those devised and recommended by Mr. Stetefeldt), for precipitating the copper and silver of the base-metal water with scrap-iron, and metallic zinc is introduced, it will precipitate the cadmium, copper, and silver. The base-metal solution is acid enough, but the addition of some sulphuric acid hastens the process.

It is more profitable to manufacture cadmium sulphide than to produce the metal. The metallic precipitate, after being washed, is boiled with dilute sulphuric acid. Cadmium dissolves, while the copper will remain

* The Mexican dollar was at the time in question, the writer believes, worth, 3s.; at present it is worth 2s. 9d.

as a sediment; and so will lead if it be present. The solution is decanted, filtered, and the cadmium precipitates as cadmium sulphide by sulphuretted hydrogen. Cadmium sulphide is a brilliant and valuable paint. Experiments on a large scale showed that from 2 to 3 lbs. of cadmium sulphide could be precipitated from the base-metal solution derived from a ton of ore.

The best orange-yellow of cadmium is obtained by precipitating with sodium sulphide, but the solution must first be made alkaline with caustic soda.

Mr. F. M. Watson in a letter to *The Engineering and Mining Journal*, New York, of April 8th, 1893,* speaking of the Russel process at Sombrerete, Zacatecas, says :—The ore runs roughly 10 per cent. of blende, 10 per cent. of galena, 30 per cent. of iron pyrites, and the rest mostly quartz containing on an average about 20 per cent. of sulphur. The old reverberatory furnaces used for roasting (taking six months' average) showed a roasting chlorination of 88·5 per cent., but the loss from volatilization was very heavy (the last six months' average being 16·6 per cent., and before it had risen to as high as 24 per cent). By considering the two losses Mr. Watson therefore places their efficiency at 73·8 per cent. The extraction of sulphides during this period was about 73 per cent., and the total cost, including grinding, was about 8 dollars, Mexican currency, per ton, crushing with rolls. He advocates using stamps in place of them. The ordinary Stetefeldt furnace also does not appear to have been more successful in this case than with the San Francisco del Oro ore, but Mr. Watson considers that it might be modified to give good results, and he considers that with a remodelled mill 85 per cent of the raw ore value could be extracted at a cost of not more than 6 dollars, Mexican currency, per ton.

Trough-lixiviation.

This system is a continuous one—a modification of tank-lixiviation—the chemical reactions are the same, but the time of leaching is enormously shortened, and the manipulations are simpler and more labour-saving. It is particularly adapted for large works, and for ore which on account of lead requires a long leaching time.

While the silver from all the other silver-bearing minerals can be easily and quickly extracted as chloride, that portion contained in the galena dissolves very slowly. The larger portion of the silver may be in fact extracted in a few hours, while the remainder will take days. The

* Page 316.

roasted del Oro ore, for example, filters quickly, but if the ore contain 10 to 11 per cent. of lead, the silver-leaching time is 4 days, and if it contain 15 to 17 per cent. of lead the time is increased to 5 or 6 days.

A description of trough-lixiviation is given by Mr. Hofmann in *The Engineering and Mining Journal*, New York, of September 10th, 1887, November 26th, 1887, and March 16th, 1889,* from which these particulars are taken. Silver chloride contained in roasted ore almost instantly dissolves if rapidly brought in contact with a proper volume of moving sodium hyposulphite solution. No more than ¾ to 1½ minutes are required, and it is rather the quantity or volume than the concentrated state of the liquid solvent, that produces this effect; it is in fact the principle upon which the process depends.

In the tanks the amount of solution which comes in contact with the ore is regulated by the latter's filtering capacity. The operator can slightly increase the speed of filtering and so increase the quantity of solution used, by producing a vacuum under the filter bottom, but he cannot produce and maintain at will a certain favourable proportion of ore and solvent, which is of such importance for a quick and thorough extraction. If, however, the ore be introduced into a running stream of the solvent, the operator has it in his power to produce and maintain any desired proportion.

Instead of charging the ore in tanks, and permitting the solvent to filter through, it is dissolved outside the tanks in troughs, while the ore is moving in and with the stream of solvent, and the tanks are only used for the separation of the solid from the liquid. The ore from the furnaces is dumped into bins, from which it is mechanically and evenly charged into a perpendicular square tube 12 by 12 and about 2 feet in height, which is crossed by several sprays or sheets of water, closing it near the top and bottom. The dust caused by the contact of hot ore and water is absorbed by the top and bottom sheet of water, and cannot escape, while the steam generated is condensed.

The pulp, when leaving the perpendicular tube, enters a grinding machine of similar construction to the German kegelmühle, an arrangement which answers better than an agitator, which was first proposed for the purpose of mixing the ore with water (to prevent it being carried down the trough in bulk), and to mash and grind up any lumps it contains. The mill discharges the pulp into the base-metal leaching-trough of triangular section, which leads to the settling-tanks.

*Pages 185, 393, and 255 respectively.

By this system, the whole cooling-floor manipulations are avoided, likewise the filling of cars on the cooling-floor and the transportation of the moistened ore to the tanks, also the shovelling out of the tailings from the tanks, which is a great saving of manual labour, without involving much extra machinery. It requires two men per shift to bring the ore from the furnaces to the bins, one man in the base-metal department, and one man and a helper in the silver-leaching department. The stirring of the silver solution in precipitating is done by mechanical stirrers, which do excellent work.

The rapidity of the extraction requires quicker circulation of the solution entailing the use of a larger pump; but Mr. Hofmann considers the extra expense of this a very small drawback. The hot ore warms the water for base-metal leaching, without incurring extra expense or labour.

The proportion of water depends on the amount of base-metal chlorides contained in the ore, and has to be so regulated that the resulting base-metal solution is too dilute to dissolve silver chloride. This is easily arranged, by gradually increasing the stream of water while maintaining the same supply of ore, and testing the resulting solution for silver.

The length of the trough depends on the character of the ore, but 150 feet from the mill to the first tank will, in most cases, be sufficient. The inclination should not be less than $\frac{3}{4}$ inch to the foot. The trough can be arranged in zigzag, but has to lead over all the tanks.

In the line of the trough above each tank there is a square box, 14 by 14 by 10 inches, the bottom of which is provided with a plug-hole in order to permit of any desired tank being charged.

The tanks have in the centre of the bottom a sluice-hole 6 inches in diameter, to which is attached to the outside by means of a flange a short piece of cast-iron pipe of about the same diameter. This pipe is like a gas-pipe elbow, and can be tightly closed by pressing a rubber gasket against the outside flange. This valve is worked from a platform alongside the tanks. This pipe must be well coated with asphaltum varnish. Around the sluice or discharge-hole the filter is arranged in funnel-shape, having an inclination of $\frac{3}{4}$ inch to the foot. The filter-cloth must be well fastened round the outer and inner circle. The central position of the discharge-opening and funnel-shape of the filter-bottom permit of quick and perfect sluicing. A tank with a level filter-bottom, if large, can be sluiced clean through a side gate. The space below the filter is provided with the usual outlet-pipe. Close under the filter is inserted from outside a $\frac{3}{4}$ inch pipe which, connected with 1 inch hose, reaches the rim of the tank for the escape of the air. Before the tank is

used for operation, a wired rubber-hose is connected with both the water and the solution-pipes, and so placed from above that the lower or outlet end enters the discharge-pipe through the central hole in the bottom. The object of this hose is to inject a stream of solution at the bottom of the ore (in the outlet-pipe), when the tank is to be sluiced out for silver leaching.

The stream undermines the tightly-packed ore which gradually caves in, until a funnel-shaped opening is made through its depths. Then one or more streams are allowed to play on top of the ore until it is all sluiced out. The hose in the discharge-pipe—which must be stiff in order not to be flattened—is allowed to remain, in order to prevent the discharge-hole from being clogged by a too sudden rush of ore. Before starting the operation the discharge-pipe is filled with water through the central hose, the lower end being covered, in order to keep it filled with water to prevent obstruction from the ore.

The tanks are placed on the same level in two rows close together, and are connected by pipes in such a way as to permit of any desired tank being disconnected without disturbing the communications of the others. The connecting-pipes are also on a level, placed a few inches below the rim of the tanks, and well coated with asphaltum varnish. Their diameter depends on the capacity of the works. Each tank is provided with an outlet (on the level of the connecting-pipes), which discharges below the bottom into the base-metal trough, the latter being connected with the outlet-hose from under the filter-bottom. The connecting-pipes and upper outlets can be closed with wooden plugs from the inside of the tanks.

Base-metal Leaching.—At starting, the stream of roasted ore and water, after having passed through the whole length of the base-metal leach trough, is allowed to enter the tank by opening the hole in the bottom of the small square box which intersects the trough above the tank. The ore will gradually fill the first tank, while the base-metal solution after reaching the level of the connecting-pipe will flow into the next tank, and when this is filled, into the next tank, and so on through all the tanks until it finds the outlet of the last tank, through which it will discharge into the base-metal trough. The motion of the solution in each tank is so slow that, by the time it reaches the last outlet, it is clear. A board, placed edgeways a few inches below the surface and across the tank, will prevent the formation of a diametrical stream from one pipe to another, and greatly assist in clearing the solution. If the proper proportions of water and ore have been used, the base-metal

solution will not contain any silver, and can be allowed to run to waste. When the first tank is filled with ore, the connecting-pipe is closed, and stream of pulp transferred into the second tank, all other connexions remaining unchanged. The outlet-hose below the filter-bottom of the first tank is now opened, and the solution still contained in it is allowed to drain off. As soon as it disappears below the surface of the ore some clear water is added to force out the remainder of the base-metal solution. The first tank is now ready for silver leaching.

Silver Leaching.—In their passage through the trough from the mill to the tanks, all the soluble base-metal chlorides have dissolved, and the charge of the first tank having been treated, as above described, is now ready for silver leaching. The rubber gasket of the discharge-pipe is pulled back, and sodium-hyposulphite solution is turned on through the central hole, and the whole charge sluiced out. Ore and hyposulphite solution discharge into a trough under the first and last tanks, which are connected with the silver-leach trough. At first, before other streams of hyposulphite solution can be played on the surface of the ore, the pulp is diluted by an extra stream in the silver-leach trough. Ore and solution now pass through a trough, not less than 150 feet long, to a similarly arranged set of tanks.

When the pulp reaches the first tank, the solution has dissolved all the silver chloride, and the sand drops as tailings into the tank. The silver solution has also to pass through all the tanks and leaves the last one clear. It is conveyed to the precipitation-tanks. When one tank is filled with tailings, it is disconnected from the others, and the pulp allowed to flow into the next one. The outlet-pipe under the filter-bottom is opened, the remaining silver solution allowed to drain off, and the part retained by absorption displaced by water. Where water is scarce, the base-metal solution can be accumulated in the outside storage-tank and used for sluicing the tailings. The proportion of solution and ore to be used depends on the nature of the ore, and has to be ascertained for each kind. Ore containing lead requires most solution. It is best to determine the proportion in weights, as it makes it easy to calculate the required capacity of the different tanks, pipes, and pumps, for a given capacity of mill.

Mr. Hofmann's tests, he states, show that the minimum quantity of solution required is 3 to 5 times the weight of the ore, and the maximum 18 to 20 times. The weight of the solution is, for convenience, taken as equal to water, 1 cubic foot = 62·5 lbs. In most cases 10 parts of solution to 1 of ore will be sufficient. Assuming this proportion, and

taking 1 cubic foot solution as 7·5 gallons, and 1 gallon as 8·33 lbs., we find that in order to lixiviate 40 tons per day, it will require 3,200 cubic feet or 24,000 gallons of solution to be daily pumped to the upper reservoir and circulated. This, theoretically, would require a pumping capacity of 16·66 gallons per minute, and precipitating facilities for 1,000 gallons per hour. By erecting a temporary trough of not less than 150 feet in length, and a mixing-box at the upper end, the required proportions of solution and ore can readily be determined by a series of experiments. The samples have to be taken at the lowest end of the trough, while the pulp is dropping into the receiving-tank. The vessel with which the sample is caught must be large enough to receive the whole stream during the time the sample is caught. Sufficient time must be allowed for the ore to settle. The clear solution is carefully decanted, and the sediment placed on a filter, washed well with water, dried, and assayed. Before subjecting it to the solution-test, the ore has, as a matter of course, to be treated with water to remove the base-metal chlorides.

Mr. Hofmann states the advantages of trough-lixiviation as follows:—

At first sight it would appear that very dilute silver solutions are obtained, and have to be treated in the precipitation-tanks. This, however, is not the case. The resulting silver solution maintains a uniform strength in silver and other soluble metal salts, and is of about the same strength as the whole filtrate of one charge in tank-leaching would be, if accumulated in one precipitation-tank. In tank-lixiviation very strong solutions are formed at the beginning, and very dilute ones towards the end, and as it frequently happens that a tank-full of such very dilute solutions has to be precipitated by itself; a uniform strength is in fact by far preferable.

This method of lixiviation allows the operator to bring the ore in sudden contact with any desired quantity of the solvent, and offers the means in this way, of overcoming some very annoying chemical and mechanical difficulties encountered in other lixiviation processes. Lead sulphate reduces the dissolving energy of sodium hyposulphite for silver, and this is why the lixiviation of lead-bearing ores is so exceedingly slow and requires such an extensive plant.

In the ordinary lixiviation, the solution becomes more saturated with lead sulphate as it descends through the ore and loses proportionately its dissolving energy. As the solubility of the lead sulphate increases with the concentration of the solution, a stronger solution does not hasten the process; but if the ore be brought rapidly into contact with a large volume of hyposulphite solution, the latter retains enough of its dissolving

energy to produce a quick extraction. The presence of lead sulphate therefore does not retard trough-lixiviation, it merely entails the use of larger quantities of solvent. The rapidity with which the silver dissolves, also materially lessens the unfavourable influence of caustic lime. The possibility of bringing the ore into contact with any desired quantity of the solvent has the further advantage of preventing the base-metal solution from dissolving silver. The resulting solution need only be sufficiently dilute. It does not take much more water than in the ordinary leaching, when the bulk of the silver is dissolved by the base-metal chlorides, while the solution is very concentrated. Shortly after the beginning, no more silver is dissolved; if, therefore, the charge of ore could have been brought at once into contact with the whole quantity of water used in washing, Mr. Hofmann thinks much less silver would go into solution. Clayey ores and flue-dust may be successfully treated by trough-lixiviation regardless of their filtering property.

In ordinary leaching, each tank-charge is in a different stage of the process, and this necessitates great care and attention and keeping a separate record of each tank. In trough-lixiviation, the operation being continuous, attention need only be concentrated on one tank in each department to prevent it from being overcharged. Although the process is divided into two departments (the base-metal and silver leaching) the manipulation is materially simplified, the tanks being charged automatically by the stream of the respective solutions, and the costly handling in cars is obviated. Leaching works built on this plan will require more grade, and it will be preferable to have the roasting and base-metal leaching done in the main building; and the silver settling and precipitation-tanks, with all the other apparatus required for the final treatment of the precipitate, in a separate department lower down; the two being connected by the silver-leach trough.

The principle of this method can be tested in the laboratory by introducing 20 grammes of roasted ore, which has previously been washed into a graduated cylinder of 1,000 cubic centimetres, in which is contained 200 cubic centimetres of sodium hyposulphite solution. The top of the cylinder has to be tightly closed with the palm of the hand, and the glass brought into a horizontal position and oscillated, in order to make the ore and solution pass quickly from one end to the other to imitate the current in a trough. After the oscillating motion has been continued for about 2 minutes the contents of the cylinder are emptied into a filter, and washed with water to displace the silver solution from the sand and filter-paper.

It will be found that all the extractable silver has been dissolved, and the residues will correspond with those obtained in the regular chlorination assay. If the result of a first trial be not satisfactory, it is due to a fault in the quantity of solution, more of which must be used in the next test, and so on, till the results are accurate and the required proportions of solution and ore are ascertained.

Trough-lixiviation was first used in the mill of the North Mexican Mining Co., Cusihuriachic, Chihuahua, Mexico. This mill was originally arranged for tank-lixiviation, but the ore hardened like cement, and did not permit any solution to pass through. Trough-lixiviation, however, enabled the ore to be worked rapidly, and the resulting tailings agreed with the chlorination test.

Mr. Hofmann gives a number of experiments and details with regard to trough-lixiviation in *The Engineering and Mining Journal*, New York, of March 16th, 1889,* and in the *Transactions* of the *American Institution of Mining Engineers*, February, 1888.† These cannot, however, be reproduced here in detail. As the saving in time is, however, one of the chief claims of the process, the writer may instance some comparative figures Mr. Hofmann gives, comparing it with tank-lixiviation treating del Oro ore :—

Time taken in tank-lixiviation :—

	Hours.
Charging	3
Base-metal leaching	8
Expelling the water by solution	1
Silver leaching	96
Expelling the solution by water	1½
Total	109½

Time taken in trough-lixiviation :—

	Hrs.	Min.
Base-metal leaching and filling the tank	3	6
To drain the wash-water from the top of ore ...	0	34
To expel the base-metal solution by water ...	3	25
To expel the water by hyposulphite solution ...	1	20
Silver leaching (sluicing with solution)	3	36
Draining solution from top of ore	0	34
Expelling the solution by water	2	30
Total	15	5

Treating the del Oro ore by trough-lixiviation, Mr. Hofmann found in silver leaching that, using a solution with a strength of 0·50 ounce per ton, the best results were obtained with the proportion of 1 of ore to 3·4 of solution or 108·8 cubic feet or 816 gallons of solution in circulation

* Page 255. † Page 662.

per ton of ore treated. But in trough-lixiviation, 658 cubic feet, or 4,935 gallons of solution were required per ton. The proportion of 1 to 3·4 gave tailings carrying only 3·59 ounces per ton. In evidence of the statement that by producing a sufficiently dilute base-metal solution it will not contain any silver and may be allowed to run to waste, Mr. Hofmann cites the following test :—One litre of the 702 gallons of base-metal solution was precipitated with calcium sulphide. The precipitate, after fluxing and treating like an ordinary ore assay, returned no more than 0·0002 gramme of fine silver, 702 grammes will therefore contain 0·532 gramme, which represented the total amount of silver dissolved from 8·39 tons of ore, or 0·06 gramme or 0·002 ounce of silver per ton, which is practically nothing.

Other points mentioned are, while in tank-lixiviation the rate of filtration was 8½ inches per hour, in trough-lixiviation it was 12 inches per hour. The number of tanks required in trough-lixiviation is much less than in tank-lixiviation, and while the roasted sulphides obtained from the trough-process contained 20·9 per cent. of fine silver, those obtained from the tank-process during the same week and from the same lot of ore only contained 17 per cent. fine silver.

Some of the most interesting figures may be summed up as follows :—

	In Tank-lixiviation.	In Trough-lixiviation.
Quantity of water required for base-metal leaching, including sluicing, per ton...	943 gals.	1,129 gals.
Quantity of hyposulphite solution which has to circulate for each ton of ore	4,935 „	816 „
Time required to treat one tank-charge of ore ...	109 h. 30 m.	15 h. 5 m.
Total quantity of water required for 100 tons of ore per day	94,300 gals.	112,900 gals.
Total quantity of hyposulphite solution per day to work 100 tons	493,500 „	81,600 „
Loss of silver in base-metal leaching per ton of ore...	0·25 oz.	trace.
Extraction of silver in both methods the same.		

In connexion with the Hofmann gold-and-silver chlorination process, which has been mentioned in the earlier part of this paper, it is to be noted that after the silver has been extracted, the solution of hyposulphite used for leaching is allowed to run out till it disappears under the surface of the ore, when clear water is introduced, in order to displace the solution. The desilverized ore must then be removed from the tank to a dry-kiln, where it is left till the surplus water has evaporated. After this it is charged back into the tank still moist.

This second handling and drying cannot be dispensed with, as the ore after leaching is too wet to permit of the free passage of chlorine. If the

ore be cupriferous, much copper will be carried out with the gold solution (after it has been chlorinated), colouring it green.

An application of the leaching process to the pan-amalgamation of very base ores, which has been invented by Mr. Kustel, deserves notice. He states that it is applicable to silver ores, like those of Flint, Idaho, containing base-metals, and also to auriferous copper ores, which by their nature require roasting, as the amalgamation of gold is very much obstructed by the presence of copper salts. If there be soluble chloride of silver in the roasted ore, and besides this, soluble chlorides of copper, lead, antimony, and zinc, they will all, as a matter of course, be decomposed and amalgamated.

All take part in consuming and flouring the quicksilver and in destroying the pan, such a combination as the above hindering the amalgamation of the silver and gold. The base-metal chlorides being soluble in water, while the chloride of silver is not, it is a simple expedient to dissolve out these salts by leaching with hot water, and thus remove them from the ore, prior to amalgamation. The ore being thus divested of its rebellious features, gives excellent results in the pans.

Lixiviation *versus* Amalgamation.

In comparing the Russel process with amalgamation, Mr. Stetefeldt* enumerates the principal points in favour of lixiviation as follows :—

1. In amalgamation, the coarseness of crushing, without considering the question of roasting, is limited by the capacity of the settler to work off coarse sands without loss of quicksilver. In lixiviation pulverizing as coarse as possible is desirable.

   The limit of coarseness depends on the character of the ore, and principally upon the manner in which the silver-bearing minerals are distributed in the gangue.
2. The original cost of the lixiviation-plant is much lower than that of pans and settlers, and a further saving is effected by a reduction in the size of engines and boilers.
3. In amalgamation the pans and settlers consume not less than $1\frac{1}{3}$ horse-power per ton of ore. The power for pumping solutions, etc., in the lixiviation process, is merely nominal.
4. In large mills the quantity of quicksilver in solution represents a capital of 30,000 to 40,000 dollars (£6,250 to £8,333 6s. 8d.), while the stock of chemicals required for lixiviation costs less than one-tenth of this amount.

* *The Lixiviation of Silver Ores*, page 4.

5. With the Russel improvements the percentage of silver extracted by lixiviation is in most cases higher than by amalgamation.
6. Lixiviation by the Russel process requires a less careful chloridizing-roasting, and in consequence a lower percentage of salt may be used in roasting.
7. Ores that can be successfully treated by raw amalgamation give often better results by lixiviation with extra solution.
8. The value of the quicksilver lost, and cost in wear and tear of the pans and settlers amounts to more than that of the chemicals consumed in the lixiviation process.
9. The lixiviation process permits of the extraction of copper and lead as valuable bye-products.
10. Amalgamation is invariably injurious to the labourers' health.
11. Where gold-bearing silver-ores have been roasted with salt, lixiviation extracts in many cases more gold than amalgamation.

The disadvantages of lixiviation as compared with amalgamation are :—

1. Lixiviation requires more chemical knowledge and a more careful supervision of the operations.
2. The handling of large volumes of solutions is a disadvantage common to all humid processes.
3. There is more danger of losing silver by careless manipulation, and by leakage of badly constructed plant.
4. In the lixiviation process the precious metals are obtained in the form of sulphides. The conversion of the latter into bullion, requires more skill and is more expensive than the handling of amalgam.

The chemistry of the process has been most ably discussed and dealt with by Mr. Stetefeldt and Mr. Ellsworth Daggett,* and treated of in the *Report of the Californian State Mineralogist*, 1888, which contains an independent article upon the hydro-metallurgy of silver, as well as a critical review of the Russel process by Mr. C. H. Aaron.†

The writer gathers from these various sources that it is with roasted ores which have an alkaline reaction caused by the presence of caustic lime (as, for example, the alkaline arsenical ores of Las Yedras, Sinaloa, Mexico,) that the Russel process, as compared with the ordinary Van Patera process (leaving the Kiss process out of the question), has

* *Trans. Am. Inst. Min. Eng.*, vol. xvi., page 362. † Page 832.

achieved its greatest success, but it is possible that it may meet with a formidable rival in this province in the new departure of pyritic smelting.*

Looking at a number of examples that are given it would appear that the extraction in the mill by the Patera process shows a difference in certain instances, in favour of the Russel process, of 22 to 30 per cent. of the silver extracted.

A table given by Mr. C. H. Aaron also shows the results of a series of competitive mill-runs, extending over eight months at the Ontario Mill (on Ontario ore), in which the percentage extracted by the Russel process varied from 84·7 to 93·9 per cent. (or on the average about 91·17 per cent.) as against 65·3 to 84·9 per cent. (or on the average 77·4 per cent.) extracted by amalgamation, treating ore which ran about 47 ozs. per ton roasted. The tailings from the amalgamation still carried 10·76 ozs. per ton on the average, whilst the lixiviation-tailings only ran 4·3 ozs.

Experiments at the same works from November, 1887, to January, 1888, on an ore which ran 43·76 ozs. in silver, showed that 82·1 per cent. was extracted by amalgamation, as against 91·5 per cent. by the Russel process.

On the whole, however, taking the results of five different mills it would seem that whilst amalgamation gave an average of 80·7 per cent., the Russel process averaged 89·4 per cent., or 8·7 per cent. above the results of amalgamation, and 4·8 per cent. better than could be extracted with ordinary solution making the latter tests in the laboratory.

It is therefore to be inferred that, in the case of certain ores which have been specified, the Russel process extracts more silver than amalgamation, but it is to be also noted that in the case of four of the mills taken as examples, the percentage extracted by amalgamation is certainly 10 per cent. less than in many mills where roasting-milling is practised, and therefore cannot be altogether regarded as typical of the capabilities of the latter process in certain cases.

The ore, for example, of the Ontario mine is very base, containing, as before pointed out, zinc, lead, and silver sulphides, as well as chlorides, in a quartz gangue, and therefore the comparative success of the process in this instance, even from a purely chemical standpoint, scarcely foreshadows it as a universal fact.

Next, as regards capital outlay, Mr. Stetefeldt gives the cost of a plant for dry-crushing and roasting 80 tons a day as follows :—

* It is to be regretted that more details of the cost and working of this process are not at present available.

	Weight in Lbs	Dollars.	£	s.	d.
Buck-eye engine, 72 to 121 horse-power	15,000 ...	1,775·00			
Knowles, feed-pump, No. 3	640 ...	275·00			
Heater ... ... / Air-pump and condenser	2,000 ...	175·00 / 450·00			
Two Krom rock-breakers	12,000 ...	1,500·00			
Three double-shelf dry kilns	70,000 ...	2,700·00			
Two sets Krom 26 inches rolls	29,000 ...	4,500·00			
Two Krom screens	1,200 ...	800·00			
Stetefeldt furnace	49,000 ...	3,000·00			
Hoist, with cage and safety-catches ...	2,000 ...	600·00			
Shaftings, bearings, pulleys, wire-rope transmission	18,000 ...	1,300·00			
Electric light plant, with separate engine for dynamos	... ...	1,700·00			
Belting	... ...	1,000·00			
Elevators, conveyors, feeders, hoppers, cone-pulley	... ...	1,000·00			
		20,775·00	= 4,328	2	6
Add the cost of four 40 horse-power high-pressure boilers	... ...	...	...		
Add the cost of the lixiviation-plant proper,* with a capacity of 80 to 170 tons of roasted and raw ore, if the tailings are removed by sluicing; 70 to 140 tons, if the tails are to be shovelled	... ...	6,772·70	... 1,310	19	7
Add the cost of painting vats and tanks with three coats of white paint, say, 746 lbs. of white lead and 41·7 gals. oil, occupying one man 34 days to lay on	... ...	...	... ...		
Add the cost of plant for refining sulphides by the humid process estimated (erected) at	... ...	5,000·00	... 1,041	13	4
Add the cost of grading and foundation to the mill structure and plant it contains (which it is only possible to figure in accurately by knowing the exact quantities of material and the time of skilled and ordinary workmen employed on the job)	... ...	...	... ...		

* This includes finished lumber and hoops for 6 lixiviation-tanks, 3 storage-tanks, and 6 precipitating-tanks, 1 sulphide storage-tank, 2 solution-sumps, cast and wrought-iron fixtures for vats and tanks, pipes, valves, steam-hose, 1 sodium sulphide mixing and 2 storage-tanks, 8 Korting ejectors, No. 4 Knowles fire-pump, (size A), for sluicing tails; Knowle plunger-pump for pumping solutions, 1 Johnson 18 inches filter-press, 1 Johnson pressure-tank, and a Knowles feed-pump, No. 2, for boiler.

Add finally freight and transport charges, depending on the locality, and it will be found that the total cost is not so far short of a pan-amalgamation plant of the same capacity as might be supposed ... ...

Now, let us glance at the working expenses of running a lixiviation-plant which are given by Mr. Stetefeldt for a mill of 80 tons capacity per day:—

	Cords of Wood.	Tons of Coal.
Fuel, engine and boilers*	5 to 6 ...	2½ to 3
Boiler lixiviating-building	3 ...	1½
Dry kiln ore and salt	3½ ...	1¾
Stetefeldt furnace, depending on quality of wood and character of ore	6½ to 8½	
Sulphide refinery for roasting, refining, and melting bars in a reverberatory furnace	1	
Total ...	19 to 22	

The cost of labour per day would be as follows:—

	No. of Workmen.		Hours of Work.
Engine and boilers ...	2	engineers	12
„ „ ...	2	firemen	12
Ore-house	2	rock-breakers	10
„	2	wheeling ore	10
Dry kilns	4	bringing ore from ore-house and charging kilns	12
„	2	discharging kilns	12
„	2	firemen also attending salt-kilns and assisting rolls	12
Rolls	2	machinists	12
Stetefeldt furnace ...	3	firemen	8
Cooling-floor	12	discharging, cooling, and charging vats ...	8
Lixiviations	2	lixiviators	12
„	3	helpers	12
„	2	precipitators	12
„	2	firemen on boilers, pumps, and pipes	12
„	2	handling sulphides	12
Refinery	2	roasters and refiners	10
Mill	1	night foreman	12
„	1	machinist	10
„	1	carpenter	10
„	1	blacksmith	10
„	4	general helps	12
„	1	assayers' help	12
„	1	blacksmith's help	10
„	1	electrician	—
„	1	bookkeeper	—
„	1	assayer	—
„	1	chief metallurgist	—
„	½	chief mining engineer	—
„	2	men and team hauling wood	—
Total ...	62½		

* It is assumed that 1 cord of wood is equal in effect to 1,000 lbs. of coal.

The daily consumption of chemicals is as follows:—

	Per Ton of Ore. Lbs.
Copper sulphate	1·8 to 9·6
Sodium hyposulphite	1·5 to 7
Caustic soda	1·4 to 7·75
Sulphur	0·9 to 5
Sulphuric acid	0·25 to 1·80

The water required is from 9 to 55 cubic feet per ton of ore, exclusive of that used for sluicing tailings.

The cost of wear and tear to cover all ordinary breakages, and wear and tear of machinery, boilers, screens, dry kilns, furnaces, lixiviation-plant, and refinery, including lubricants, electric light, and sundries, is about £7 5s. 10d. per day. For plants of small capacity the expenses per ton of ore, for labour, are, of course, materially increased. To the above must be added insurance, taxes, interest, legal expenses, etc., and amortization on the capital invested in the mill.

The total expenses at Cusi, Mexico, including refining, are said to have been £2 10s. 4d. per ton, using the Russel process.

For roasted ore at the Sierra Grande mill, Lake Valley, the total mill expenses for 60 tons per day were estimated by the general manager, Mr. Hadley, at 19s. 4d. per ton.

At Parral, Mexico, the total expenses for the treatment of tailings (from ore which had originally been roasted and lixiviated by the ordinary process) were 8s. 9d. per ton for 40 tons per day. Treating roasted ores at the rate of 10 tons per day the total expenses were £1 18s. 1½d. per ton.

The cost of raw-leaching tailings at Silver Reef, Utah, treating 25 tons per day (the tailings assaying 6½ to 9½ ozs. of silver but no gold), is given by Mr. Egleston* as 6s. 10½d. per ton, the percentage extracted averaging 55 to 60 per cent. The presence of copper carbonate in these ores caused the sulphides to have a low percentage of silver (3,560 ozs. per ton) and a high tenour of copper.

Treating tailings from amalgamation-works at Silver City, concentrated up to 30 ozs. per ton silver, it is asserted that the Van Patera process only extracted 38 per cent. against 72·4 obtained by the Russel process.

Mr. Stetefeldt states that in a well-constructed lixiviation-mill the total expenses for treating 75 tons of raw ore per day should not exceed 12s. 6d. per ton, and under favourable circumstances should fall as low as 10s. 5d. per ton, particularly if the crushing were done by rolls instead of stamps.

* *Metallurgy of Silver*, page 532.

The chemicals required to be carried in stock for a lixiviation-plant of 80 tons daily capacity are estimated in an assumed case as amounting to about £878 0s. 2d., to last for 60 days, including sodium hyposulphite in stock solution.

Mr. Stetefeldt, page 213, says: "The Ontario mill expenses are about £2 14s. 2d.* per ton, those for lixiviation would be about £1 0s. 10d. to £1 5s. less, or a difference in favour of the Russel process of £1 0s. 10d. to £1 5s. in expenses, which, together with the additional extraction of 17s. 1½d., would make a total net difference of £1 17s. 6d. to £2 1s. 8d. per ton in favour of lixiviation."

As already remarked, however, in this the favourite instance taken for comparison of the two processes, the mill extraction by amalgamation is not nearly so high as it might be, and in the same way the cost, is nearly 12s. 6d. above the average of many amalgamation-works. Compare, for example, the expenses of mill B of the Granite Mountain Co. (43 stamps) for the year ending July 3rd, 1891, and it will be found that the milling charges did not exceed £2 4s. 8d. per dry ton, treating 19,463 tons dry, whilst in the case of the Elkhorn mill, which treated 11,645 tons, the cost was only £1 18s. 5¼d.

If one compares the amount of labour required in a lixiviation and an ordinary amalgamation-mill in places where labour is dear, save in some exceptional cases, lixiviation must be the more costly process of the two, and it is in fact in countries like Mexico, where the conditions in this and other respects are favourable that it will find its widest adoption, until the labour costs common to all processes of the kind can be reduced considerably.

In confirmation of this statement the writer may cite the Geddes and Bertrand mill, Nevada† (an ordinary lixiviation-plant dealing with 50 to 60 tons per day), the staff of which, when running, consisted of 60 men, whose wages amounted to £30 6s. 3d. per day.‡ The cost of lixiviation milling (using rolls) came to £1 7s. 1d. per ton, and 13s. 8½d to £1 0s. 2d. in silver was still left in the tub-tailings.

Whilst leaching is doubtless applicable to certain ores in certain

* Mr. Egleston gives the actual running expenses, treating Ontario ore by amalgamation on a production of 50 tons per day as £3 2s. 7d., agreeing with Mr. Rothwell's figures. These works seem in fact to have employed an exceptionally large staff for a 40 stamp mill, viz., 66 to 72 men.

† The failure of amalgamation appears to have been due to the presence of antimoniate of lead in the ore.

‡ Egleston, "Leaching Gold and Silver Ores in the West." *Trans. Am. Inst. Min. Eng.*, vol. xii., page 40.

localities, it is always a process which involves highly skilled superintendence and chemical supervision, otherwise very serious losses may be incurred in roasting, leaching, and precipitating the metals, and in this way alone the possible profit to be gained by employing it may easily be converted into a positive loss.

Treatment of the Sulphides obtained from the Russel Process.

One of the great drawbacks of the Russel process which has been referred to, and which is common to other lixiviation processes yielding a base precipitate, viz., the treatment of the sulphides of lead, copper, iron, silver, and gold, produced by precipitation with alkaline sulphides, is now believed to have been overcome by a process invented by Mr. Cabell Whitehead. This new mode of treatment, it is stated, can be effected at a cost of 1¼ cents per ounce of contained silver, at the same time avoiding the frequent loss of silver that other methods entailed.

Refining on a cupellation hearth (the old method which is still in use in several mills) has the disadvantage of causing large losses of silver by volatilization in the previous roasting, and the locking up and eventual loss of a portion of the silver in slags, cupel-bottoms, and matte.

From 80 per cent. to 90 per cent. only of the silver charged was obtained in the shape of bars. The rest would be in greyish-black copper-lead silver matte, which would be formed early in the operation in the pasty slags which it was impossible to get rid of, and again the precious metals had a most pernicious faculty of sinking, not only into the cupel-bottoms, but below even into the iron-pan and into the surrounding brickwork. All these products had to be treated again, or shipped direct to a smelter. If they were shipped differences in assays, and possible losses were bound to occur, as the products contained shot silver. If the sulphides were charged upon the bath without undergoing a previous roasting under the mistaken idea of preventing the loss of silver by volatilization, the same troubles occurred in even a more pronounced degree. If the company shipped its sulphides direct to smelting or refining works, it avoided the great losses of this crude process, but incurred heavy expenses and discounts, amounting to as much as 10 per cent. on an average, in the case of a Mexican mine.

At present interest centres on two new processes, the one above alluded to, invented by Mr. Cabell Whitehead. This process is confined to the treatment of the sulphides as they are found in the precipitation tubs of a lixiviation plant. Its details have not yet been made public, but it is

stated that its success has been experimentally demonstrated, and the refining department of the Marsac mill of the Daly Mining Co., has been remodelled with a view to its adoption.

The other process aims to get at the root of the difficulty by precipitating electrolytically on zinc plates. Its success, whilst doubtful, must yet be proven before it is entitled to serious consideration.*

Examples of the Russel Process.

The careful preparation of the ore by a thorough chloridizing-roasting appears to be one of the chief points on which the success of the Russel process, in such cases as it may be applicable, turns. It does not appear to be adapted for the treatment of ores containing metallic silver, which is comparatively insoluble in cuprous hyposulphite. An account of the results of the process at two of the works where it has achieved its most successful results may be of interest.

For a description of its working at Las Yedras, Sinaloa, Mexico, the writer is indebted to an article in *The Engineering and Mining Journal*, New York, dated January 14th, 1893,† contributed by Mr. R. F. Letts. The Yedras mine of the Anglo-Mexican Mining Company is situated in the north-eastern corner of Sinaloa.

A 40 stamp mill and a lixiviation-plant to use the Patera or Kiss processes was erected in 1882. Poor results were obtained, and the Brückner furnaces which had been introduced were abandoned for the cruder but more satisfactory reverberatory, as the latter did not ball or agglomerate the roasted ore. It is estimated that the Patera process did not save over 65 per cent. of the silver in the roasted ore.

The following are two analyses of Yedras ore, representing the averages of the ore treated at different periods:—

	No. 1.		No. 2.
Carbonate of lime	33·78	...	46·50
Silica	15·13	...	25·00
Iron	17·33	...	9·80
Sulphur	13·31	...	12·50
Arsenic	9·82	...	2·50
Zinc	4·92	...	—
Lead...	1·78	...	—
Magnesia	2·58	...	—
Alumina	1·35	...	—

No. 1 is an analysis of the average battery sample for one month. The composition of the ore varies greatly. A couple of months later, analysis

* *The Engineering and Mining Journal*, New York, February 25th, 1893, page 169.

† Page 34.

No. 2 was made; the battery samples for several weeks showed 4 per cent. of zinc, and two months later contained a large percentage of antimony.

The results by using the Patera process were as follows:—Assay of ore, 60·67 ounces per ton; extraction in assay office, 72·09 per cent.; extraction in mill, 67·12 per cent.; total leaching time, 92 hours.

By the Russel process the results were as follows:—Assay of ore, 55·3 ounces per ton; extracted by old process in assay office, 69·94 per cent.; extracted by Russel process in assay office, 83·62 per cent.; extracted by Russel process in mill, 82·44 per cent.; leaching time, 76 hours.

It will thus be seen that the extraction by the Russel process was 15·32 per cent. higher than by the old process, and that the leaching time was 16 hours shorter. The chemicals consumed per ton of ore were as follows:—

	Old Process. Lbs.	Russel Process. Lbs.
Lime	9·7	—
Sulphur	4·7	3·6
Hyposulphite of soda	—	1·4
Bluestone	—	9·6
Caustic soda	—	5·5
Total	14·4	20·1
Cost per ton ...	$0·52 = 2s. 2d.	$2·20 = 9s. 2d.

Since these runs were made, the consumption of chemicals has been reduced to 9·14 lbs. per ton of ore and the cost to 0·82 dollars.

Owing to the great distance from the railroad, the price of chemicals in Yedras is of course considerably greater than at most places in the United States or at many localities in Mexico. The average cost of chemicals per lb. at Yedras for the last three years is as follows:—Hyposulphite of soda, 8 cents; bluestone, 10 cents; caustic soda, 9·1 cents; and sulphur, 7·1 cents. No soda ash (sodium carbonate) is used at Yedras as there is usually no lead in the ore. The total cost of all chemicals at Yedras in 1890 was 3·6 cents per ounce of silver produced. Of the copper used in the form of bluestone about 50 per cent. remains in the ore.

During the past five years the comparative efficiency of the two processes at the Yedras mill has been tested four times. The duration of each of these tests was from one to three months. Two methods were pursued in making these comparative runs. One was to divide the roasted ore equally between the two processes, running one-half the ore vats and precipitating-tanks on the old process and the other half by the Russel process, the products being kept entirely separate, and the tailings from each process thrown out.

By this method, however, there is a loss of 8 or 10 ounces per ton on all ore treated by the old leaching process, as that amount, which might be extracted by the Russel process if it were used on the same charges after the old process, remains in the tailings.

In the other method all the charges are treated first by the old process, that is, by the simple hyposulphite solution, until no more silver can be extracted, the sulphides being precipitated by themselves and kept separate. Then these same charges of ore are treated by the Russel process, that is, by cuprous hyposulphite or extra solution. The precipitates from this solution are likewise kept separate. In this way a comparison between the two processes is made without any loss, each charge of ore having the benefit of being treated by both processes before it is thrown out. In fact, this is the way all the ore is treated at Yedras, all charges being first treated by the old method, and then by the Russel process.

The first of the two comparative runs was made by Mr. Letts between the two processes in September, 1890; the test lasting a month.

An extract from his report to the Anglo-Mexican Company runs as follows:—"Our intention was to give the old process every possible show. Great care was taken to keep the precipitates separate, both at the beginning and end of the month. During the month no experiments or extra clean-up was carried out. In making the test we allowed the old process to take out all it could take, *i.e.*, we ran the vats (by the old process) so long as sodium sulphide would show the least trace of silver in the solution.

When the old process would not take any more silver out, the extra solution of the Russel process was applied, and, as in the case of the old process, was run as long as sodium sulphide showed any trace of silver."

The actual clean-ups from the two processes were as follows:—

	Ounces.
Extracted by the old leaching process	26,361·42
Additional extracted by the Russel process	5,088·76
Total produce	31,450·18
Per cent. of total produce extracted by old process ...	83·8
Per cent. of total produce extracted by Russel process	16·2

The total additional cost of the Russel process, or, in other words, the extra expense of producing 5,088·76 ounces over the cost of the old process was as follows:—Chemicals, 632·50 dollars; fuel, 21·21 dollars; extra help, etc., 155·60 dollars; total, 809·31 dollars (£166 14s. 7½d.). At the then price of silver, the 5,088·76 ounces equalled 5,852·07 dollars. Deducting the above expenses 5,042·76 dollars (£1,050 11s. 6d.) is left as the net profit per month due to the extra treatment.

Another test was carried out during the month of November, 1890. In this run the total cost of chemicals per ton was 95 cents (3s. 11½d.). Of the total number of ounces extracted, the old process took out 80·29 per cent., and the Russel process the remaining 19·71 per cent. The additional ounces of silver extracted by the Russel process over the old process were 7,653·7, or (with silver at 1·025 dollars (4s. 3¼d.) per ounce) 7,845·04 dollars, making a net profit due to the use of the Russel system of about 7,000 dollars per month.

All the tailings which have been produced at the Yedras mill, by the old process before the introduction of the Russel process in 1887, have now been treated by the latter. The tailings are brought from the old dumps where they were thrown out in former years, and are charged direct to the leaching vats without any drying, roasting, or other treatment. Like the charges of ore, they are leached with water in order to remove the small percentage of soluble salts present; this washing requiring about four hours. A small percentage of ordinary hyposulphite solution is then applied, since the volume of the extra solution is only enough to saturate the charge, and as it would become diluted to some extent with the wash-water if it followed it, the small volume of ordinary solution is interposed.

As in treating ore, this extra solution amounts to 13 cubic feet per ton. It is followed by more of the hyposulphite solution to extract any silver which has been made soluble by the extra solution, but which has not passed out of the charge with it, remaining mechanically held in the pulp.

The total quantity of tailings from the old leaching process at Yedras, which have been re-treated by the Russel process, is between 30,000 to 40,000 tons. The following table shows the results:—

Year	Ounces.	Extraction in Assay Office. Old Process. Per Cent.	Extraction in Assay Office. Russel Process. Per Cent.	Apparent Extraction in Mill. Per Cent.	Actual Extraction by Russel Process. Per Cent.
1888 ...	19·49 ...	37·40 ...	62·70 ...	60·14 ...	62·74
1889 ...	17·23 ...	32·17 ...	57·20 ...	55·95 ...	60·14
1890 ...	13·46 ...	38·48 ...	49·26 ...	48·37 ...	46·84

In the above table, apparent extraction in mill is obtained by comparing the value of the final tailings from the Russel process with the old tailings, as charged to the leaching-vats (taking also into account any soluble salts).

Actual extraction in mill is obtained by comparing the clean-up in silver with the silver actually charged to the vats. The chemicals used per ton of ore during these three years were as follows:—

Year.	Hypo-sulphite.	Blue-stone.	Caustic Soda.	Sulphur.	Total Chemicals per Ton.	Cost of Chemicals per Ton.	Ounces of Silver Extracted per Ton.
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Dollars. s. d.	
1888	2·08	6·32	5·54	3·30	16·24	1·45 6 0½	12·23
1889	1·62	5·11	3·13	2·69	12·55	1·12 4 8	9·75
1890	1·07	4·06	2·48	1·71	9·32	0·82 3 5	6·30

For a description of the Russel process at the Marsac mill, Park City, Utah, the writer is indebted to an article by Mr. W. G. Lamb in *The Engineering and Mining Journal*, New York, of December 17th, 1892.* It was started at these works on January 1st, 1889, superseding amalgamation at the end of that year. The statistics of amalgamation to which reference is made are from the Ontario mill in the same camp.

In that mill, amalgamation has been in continuous use since its start in January, 1887. As the wages and prices of fuel and supplies are the same for the two mills, a comparison of statistics is of value in determining the general efficiency and economy of the two processes, dealing, it should be added, with the same classes of ore, as the accompanying analyses indicate. The properties of the two companies, the Ontario and Daly, adjoin, and are in fact on the same vein. The equipment of the two mills and the staff employed are as follows:—Ontario, 2 rock-breakers, 2 rotatory driers, 40 ore-stamps, 10 salt-stamps, 2 Stetefeldt furnaces, 24 pans, 12 settlers, 71 mill men. Marsac, 1 rock-breaker, 2 rotatory driers, 30 ore-stamps, 5 salt-stamps, 1 Stetefeldt furnace, 6 (16½ feet) ore-vats, 8 (9 feet) precipitating-tanks, 51 mill men.

In the above connexion it is to be noted as before remarked that the labour item compared with figures given elsewhere appears unusually high at the Ontario for a 40 stamp amalgamating-battery, and correspondingly low for a lixiviation-plant of the same size, if the whole of the staff be included in both.

The analyses and values of the ore treated at the Ontario and Marsac mills for 1891 are as follows (the samples on which these analyses were made being composed of all the battery samples taken each day during the entire year):—

	Ontario.	Marsac
Silica	75·0	76·60
Zinc	5·73	5·30
Lead	1·80	3·50
Iron	2·80	1·65
Sulphur	2·23	0·70
Lime	1·76	1·32
Magnesia	0·23	trace.
Copper	0·29	0·39
Silver (ounces)	39·50	39·10
Gold (ounces)	0·044 ($0·91)	0·044 ($0·91 = 3s. 9½d.)

* Page 580.

The following table gives the crushing statistics for 1891 :—

Mill.	Ore.	Time Battery run.	Mesh of Screen.	Rate of Crushing per Day.	Rate of Crushing per Stamp, per Day.
	Tons.	Days.		Tons.	Tons.
Ontario (40 stamps)...	25,650	341·8	26	75·0	1·87
Marsac (30 stamps)...	24,214	347·0	20	70·0	2·33

The above difference in rate of crushing per stamp is probably not due entirely to difference in mesh of screen, for Ontario ore may not crush so fast as Daly, even in the same battery and with the same mesh of screen. Here again we have a matter which may affect the relative results (supposing one process replacing the other) in regard to cost.

The Ontario product in bars of bullion averaged 425 fine in silver and 0·250 fine in gold; it contained also 57·5 per cent. copper. The Daly precipitates, including those from the wash-water, but not the lead carbonate product, averaged 313 fine in silver and 0·260 fine in gold; they contained 15·3 per cent. copper.

The cost of marketing the product was 3·47 cents per ounce for Ontario bullion and 3·45 cents per ounce for Marsac sulphides. The price obtained was 97·55 cents per ounce for silver in Ontario bullion and 97 cents for that in Marsac sulphides. In the Marsac sulphides shipped, 20·67 cents per ounce was received for the contained gold, against nothing for that contained in Ontario bullion.

The following is the consumption of water, chemicals, iron, and power per ton of ore at the Ontario and Marsac mills :—

Mill.	Water.	Chemicals and Mercury. Cost.	Iron.	Power.*	Machinery Expenses.
	Cubic Feet.	Dollars.	Lbs.	Horse-power.	Dollars.
Ontario	400	1·315	5·5	108	0·31
Marsac	56	0·924	0·05	8	0·07

The consumption of chemicals has increased since 1890 owing to the adoption of hot-solutions. The production increased 2·8 per cent. however, and while the total cost was increased 7,339·98 dollars the net gain in extraction after deducting the extra cost amounted to 18,478·92 dollars.

The following are some additional details of work at the Marsac and Ontario mills in 1891, when the 30 stamp Marsac mill crushed 24,215 tons of ore through a 20 mesh screen against 25,650 tons through a 26 mesh screen at the 40 stamp Ontario mill :—

Mill.	Fuel. Per Ton. Dollars.	Salt used in Roasting. Per Cent.	Labour.† Dollars.	Extraction of Silver. Per Cent.
Ontario (cords of wood)	0·153	13·9	0·46	90·8
Marsac (tons of coal)	0·087	8·26	0·31	91·8

* This is for power for driving pans and settlers at the Ontario, and for stirring and handling solution and grinding sulphides at Marsac.

† Includes that on the pans, amalgam, and bullion at Ontario; and on vats and shipment of sulphides at Marsac.

During 1892, up to December 1st, the percentage of salt used at Ontario had been increased to 14·2 per cent., and that at Marsac 9·5 per cent. The extraction at the Ontario mill remained at 90·8 per cent., and that at the Marsac 91·9 per cent. It is probable that the extraction at the latter works would be increased if the cooling floor-space were enlarged, so as to allow the ore to cool without wetting down. This would increase the expense, it is estimated, by only 13 cents per ton.

The following is the detailed annual cost of the lixiviation department at the Marsac mill:—

	Dollars.	Dollars.	s.	d.
Labour—				
1 foreman, $5·00; 3 leachers, $4·00; 3 shovellers, $3·00; 1 pressman, $3·50; 1 labourer, $1·50	16,790·00	... 0·6934 =	2	10¾
Chemicals—	Dollars.	Dollars.		
Hyposulphite ... 152,808 lbs. at ·0362	= 5,531·64			
Bluestone ... 78,569 lbs. at ·0641	= 5,036·27			
Caustic ... 119,741 lbs. at ·0555	= 6,645·62			
Sulphur ... 80,486 lbs. at ·0257	= 2,068·49			
Soda ash ... 22,309 lbs. at ·0317	= 707·19			
	19,989·00	... 0·8255 =	3	5¼
Repairs—				
1 machinist, $4·00; materials and supplies, $2·50 per day	2,372·50	... 0·0979 =	0	4¾
Power	3,139·89	... 0·1297 =	0	6½
Assay office	2,008·82	... 0·0830 =	0	4¼
Total cost	$44,350·21	... $1·8295 =	7	7½

Figures for the following comparison of the Ontario and Marsac results for 1891 are taken from the respective reports of the Ontario and Daly mining companies:—

	Dollars.	Dollars.	£	s.	d.
Ontario—					
Cost of milling per ton	8·93				
Product expense	1·23				
		10·16 =	2	2	4
Marsac—					
Cost of milling per ton	6·27				
Product expense	1·23½				
		7·50½ =	1	11	3
Difference in favour of Marsac		2·66 =	0	11	1
Ontario—Mill extraction	91·00 %				
Marsac „	91·57 „				
Difference in favour of Marsac ...	0·57 „	= 0·21½ =	0	0	10¾
Ontario—Realized from gold	0·00				
Marsac „ „	0·63½				
Difference in favour of Marsac ...		0·63½ =	0	0	7¾
Total difference claimed in favour of Russel process		3·51 =	0	14	7½

This total difference of 3·51 dollars would, it is asserted, have made a saving of 91,057 dollars, had the Ontario ore, amounting to 25,650 tons in 1891, been treated by the Russel process. Mr. Lamb sums up by saying that to treat about the same number of tons of ore per day, of approximately the same composition at the Marsac mill, the Ontario mill requires 39 per cent. more labour, 30 per cent. more stamps, more power, twice the number of furnaces, 48 per cent. more salt, and 40 per cent. greater cost of chemicals, and yields a smaller percentage of both gold and silver than the Marsac mill using the Russel process.

Whilst this may be true, a slight difference in the composition of the ore may affect the extraction, and looking at the respective analyses given, it seems hardly fair to assume that, even in the case of Ontario and Marsac ores, the commercial results of the lixiviation treatment will be exactly the same in both cases.

A comparison of extraction by lixiviation (using the Russel process) with that by amalgamation made in 1891 at the new works of the Blue Bird mine, Montana,* showed that the extraction by amalgamation varied from 58·5 per cent. to 80 per cent., while with lixiviation it averaged 84·1 per cent. It is claimed that the extraction might have been 8 per cent. higher (judging from experience elsewhere) if the lixiviation charges had not been wetted down hot.

The analysis of the battery samples for six months showed the ore to contain :—

	Per Cent.
Silica	64·4
Sulphur	5·0
Iron	3·74
Lead	4·22
Zinc	12·8
Manganese	5·21
Copper	0·20

The cost of chemicals and quicksilver averaged 0·80 dollars for amalgamation and 0·99 for lixiviation, using 3·3 to 3·7 lbs. of hypo-sulphite of soda, 5·6 to 9 lbs. of bluestone, 3·7 to 6·7 lbs. of soda ash, 4·4 to 5 lbs. of caustic soda, and 2·5 to 3·4 lbs. of sulphur, making a total of 18·8 to 27·5 lbs. of chemicals per ton of ore treated.

The leaching vats were filled with 20 to 70 tons charges about 7½ feet deep. The first part of the wash-water was run in from below the filter, while the ore was being charged into the vat, the leaching with water being afterwards introduced from below downwards as soon as the ore

* C. A. Hoyt, *The Engineering and Mining Journal*, New York, Jan. 7th, 1893, page 8.

was charged. The base-metal leaching was followed by about 100 inches in depth of ordinary solution. This was succeeded by about 30 inches of extra solution, containing 1 per cent. of bluestone, which was allowed to stand seven to ten hours. This was again followed by 40 to 60 inches of ordinary solution, and then by 10 inches of extra solution of the same strength as before, which was allowed to stand seven to ten hours, and finally 50 to 60 inches of ordinary solution was run in and was expelled by the second wash-water.

The strength of the stock solution was 1·6 per cent. to 1·9 per cent. of hyposulphite of soda, whilst the extra solution contained in addition 1 per cent. of bluestone.

All solutions were kept at a temperature of 90 to 120 degs. Fahr., 11 to 15 per cent. of salt was used in roasting, and the ore was crushed on the average to pass a 24 mesh screen, the salt to 20 mesh.

The silver and gold were precipitated from both solutions and wash-water with sodium sulphide; and the lead by itself from the solutions by soda-ash.

Mr. H. Lang, in a letter to *The Engineering and Mining Journal*,* New York, of March 18th, 1893, remarks:—"The Russel Company summarizing their claims declare that the Russel process, both metallurgically and economically, occupies the place formerly held by:—1st, the Kiss-patera, or old leaching process; 2nd, amalgamation of silver and silver-gold ores; 3rd, smelting of dry ores, and ores averaging not over 15 per cent. lead, or such as do not contain sufficient lime or iron to make them desirable as fluxes in smelting."

Without expressing an opinion on the merits of the first two claims, Mr. Lang takes emphatic exception to the third and says:—"In no case and under no conditions can the Russel process treat basic ores as cheaply or as efficiently as can the smelting processes. With acid ores I recognize in full its advantages, but even with the most siliceous material it is questionable if the process can always compete with matting, even when silver alone is treated; and when gold, copper, and other metals are worked for, it has no chance whatever."

In 1889, a company of Oregon capitalists erected a rather complete Russel process mill at Mineral, Idaho (a quarter of a mile below the spot where a matting plant now stands) at a cost of £6,250, which was under the charge of a skilful leaching-expert, Mr. W. H. Lamb.

Mr. Lamb laboured arduously and intelligently through several months; but, in vain, the project was a failure, and the mill was shut down and sold for a tenth of its cost for other purposes.

* Page 244.

According to Mr. Lang's information, the best results reached 76 per cent., though it does not appear whether this was the extraction, apparent extraction, or simply an extra extraction. He goes on to add that the ores in question (of which only the more tractable part were sought to be worked in the Russel mill) have been bought by his firm during the last three years, and successfully treated in matting furnaces. That they have made it pay where the Russel process failed, is a sufficient answer to the assertion that the process occupies the place of smelting.

Treating the same ores, he claims to be able to do the work at one-fifth the cost, and save 20 per cent. more silver than the Russel process claimed to extract.

The high cost of treatment at these works is stated, however, to have arisen largely from faults of construction and design.

The claims of the process for the treatment of basic ores are only relatively true. The Russel Company quote an analysis of the Las Yedras ore (Sinaloa, Mexico) and say "smelting being economically and *metallurgically** out of the question." (This was said of an ore containing silica, 25 per cent.; calcite, 46 per cent.; iron, 9·8 per cent.; sulphur, 12·5 per cent.; and arsenic, 2·5 per cent.) Instead of being an unsmeltable combination, this ore is in reality the finest smelting product in the world, susceptible of being run down at one operation into a high-grade matte, and at less than the cost of the salt which is now used in roasting. And the matte can then be refined, and its total silver extracted at an additional cost per ton of original ore, not exceeding the cost of the chemicals now used in the Russel leaching. Mr. Lang bases this opinion on the data given in the valuable series of papers by Mr. Rockwell, published in *The Engineering and Mining Journal*,† New York, "On Roasting, Chloridizing, and Lixiviation at Yedras mine, Mexico." The Yedras ore is chemically nearly the same as that of an important mine near Mineral Hill, of which considerable quantities have been treated by Mr. Lang's firm, the difference being an excess of carbonate of lime in the Mexican ore. This is run down without the use of fluxes and without admixture of other ore, using 7 per cent. of coke.

Employing a furnace of special construction and by peculiar treatment of the blast, etc, the larger part of the sulphur and arsenic is burnt off, and the corresponding proportion of iron and zinc is slagged off, effecting a desirable concentration of the matte and at the same time utilizing the heat of combustion of the elements named. This is pyritic smelting, properly so-called, a branch of the larger art of matte smelting

* The italics are mine. † Vol. xlv., pages 86, 106, 159, 178, 197, 213, and 283.

The waste of silver at Las Yedras must have been prodigious. Mr. Rockwell mentions months of work in which the losses by volatilization varied from 17 to 25 per cent., the best work attainable in his time resulting in an average loss of 10 per cent. from that cause. The Russel company have recorded it at 6 or 7 per cent. The total losses now amount to about 19 per cent.

The Russel process has undoubtedly been a great improvement over the old leaching with certain classes of ore, but it is a great wonder that the management tolerated any lixiviation methods whatever.

Mr. Lang states that he holds the same opinion concerning the Aspen* and Marsac works. The composition of the Aspen ore (30,000 tons of which have been treated by the Russel process) is lead, 2·27 per cent.; silica, 21·66 per cent.; sulphate of barium, 20·92 per cent.; lime, 10·99 per cent.; magnesia, 4·24 per cent.; iron, 10·02 per cent.; zinc, 2·85 per cent.; copper, 16 per cent.; sulphur, 8·10 per cent.; and arsenic, traces.

* The probable cost of pyritic smelting in this district would be under 12s. 6d. per ton, treating not less than 100 tons per day.

THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.*

By A. G. CHARLETON.

Part IV.—Gold.

Free-milling and Grinding-milling.†

The cost of plant and of treatment of this character, in various parts of the world must now be considered, and to do so the author has drawn up Table I. hereto appended, which gives particulars of the plant and the approximate cost, as far as it can be estimated, of various well-known mills, gathered from such information as can be obtained on the subject.

Turning to Table I., it will be remarked that the cost of plant per stamp-head is least in Wales and America and greatest in Queensland, Australia, though it might cost a good deal more in out-of-the-way parts of Africa, South America, or Asia. It is stated that some of the earlier Johannesburg batteries cost £800 per stamp,‡ and the old Eureka mill, in California, cost £500 per stamp.

The low cost of the Welsh§ and American mills is partly explained by the fact that there are no heavy duties and comparatively slight freight charges to pay, which are a heavy item in other places, such as Queensland. The chief reasons, however, are that the former mills dispense with the grinding, tailings, and other plant that adds considerably to the cost of the Australian batteries, and that the prices of timber, lumber, and supplies are relatively very large in the Colonies.

For example:—Lumber in California costs about 75s. per 1,000 B.M., and in Queensland, Oregon pine ranging in size from 3 inches by 2 inches up to 9 inches by 9 inches, cost about 18s. 6d. to 21s. per 100 B.M. in 1887-8. Oregon lumber (imported) ranging in size from 8 inches by 8 inches up to 12 inches by 12 inches‖—cost about 1s. 6d. to 2s. 9d. per lineal foot in 1887-8. Redwood boards (well-seasoned) up to 18 inches in width, cost about 26s. to 28s. per 100 B.M. in 1887-8.

* *Trans. Fed. Inst.*, vol. iv., pages 233 and 351, and vol. v., page 271.

† For the treatment of gold ores.

‡ *Mining Journal*, August 27, 1891, page 977.

§ The lower wages-rates, freight-rates, and price of materials in Wales explains the difference which exists between the Welsh and the American costs.

‖ In sticks, 36 to 40 feet in length.

TABLE I.

Showing the Character and Approximate Cost of Free-milling and Grinding-milling Plant in Different Localities.

Locality, District, and Name of Mill or Company.	Nature of Mill Site.	Character of Process.	Water or Steam Power.	Rock Breakers.	Stamps.	Weight of Stamps.	Automatic Feeders.	Number and Kind of Concentrators.	Pans.	Settlers.	Other Machinery.	Approximate Cost of Milling Plant and Mill Buildings.	Cost of Accessory Plant.	Cost per Stamp.	Remarks.
						Lbs.						£	£	£	
Wales—															
Dolgelly—															
New Morgan ..	*a*	Battery amalgamation and plates with ripples.	120 H.P. turbine & 125 H.P. eng.	2	40	850	8	Embrey pattern concentrs. used in old mill; not now employed.	....	..	..	8,500*b*	..	212	10 stamps in a battery, with pulley in the middle of the cam shaft. 4½ tons of coal burnt per day.
Italy—															
Pestarena—															
Pestarena	..	....	Water power..	1	..	*c*	..	Inclined tables	....	..	*d*	....	..	..	.....
India—															
Wynaad—															
Indian Consolidated (Phœnix)	*e*	Free gold in battery, and plates, ripples, and pans.	Turbine and engine.	..	20	850	..	4 end-blow tables.	7 Wheelers ..	6	*f*	....	..	..	There are six Wheelers and four Hungarian mills in reserve.
Transvaal (So. Africa),															
Johannesburg—															
Jumpers	*e*	Free gold in battery and plates outside.	Comp. engine, 100 H.P.	4	100	780	..	20 Frues, 4 shaking tables, 2 Buddles.	2 Wheelers, 1 Berdan.	1	*r*	45,395	..	454	*v*; 4 240 × 60; 550 tons of coal burnt per month.
Dakota (U.S.A.)—															
Deadwood Gulch—															
Father de Smet ..	*g*	Inside & outside plates and mercury traps and sluice boxes.	Engine with Meyer cut-off (20 × 42).	4	80	758	16	Blankets ..	2 clean-up ..	1	*p*	20,338	653	254	*h*; 4 68 × 60 *j*; 11 cords of wood per day.
Lead City—															
Highland	*g*	Do. ..	Eng. 350 H.P., Corliss.	6*k*	120	850	24	Blankets ..	2 clean-up ..	1	*p*	51,042	..	425	*l*; 4 100 × 80; 14 cords of wood per day.
Homestake	*g*	Do. ..	Eng. 155 H.P., aut. cut-off.	5	80	850	16	Blankets ..	2 clean-up ..	1	*p*	....	..	..	*l*; 4 100 × 80; 11 cords of wood per day.
Golden Star ..	*g*	Do. ..	Eng. 300 H.P. Harris-Corliss.	6	120	850	4	Blankets ..	2 clean-up ..	1	*p*	....	..	..	*l*; 4 120 × 88½; 14 cords of wood per day.

TABLE I.—*Continued.*

Locality, District, and Name of Mill or Company.	Nature of Mill Site.	Character of Process.	Water or Steam Power.	Rock Breakers.	Stamps.	Weight of Stamps.	Automatic Feeders.	Number and Kind of Concentrators.	Pans.	Settlers.	Other Machinery.	Approximate Cost of Milling Plant and Mill Buildings.	Cost of Accessory Plant.	Cost per Stamp.	Remarks.
						Lbs.						£	£	£	
Alaska (U.S.A.)— Douglas Island— Alaska-Treadwell ..	*g*	Inside & outside plates and mercury traps and sluice boxes.	6 ft. Knight and 6 ft. Pelton.	6	240	900	48	96 Frues ..	....	..	..	....	..	..	Concentrates chlorinated.
California (U.S.A.)— Plumas County— Plumas-Eureka ..	*m*	Inside & outside plates & silver sluices.	8 ft. Knight and 75 H.P. engine.	3	60	850	12	20 Hendys, 5 Pattons, 3 Duncans.	2 Patton pans, 1 cln.-up bar. rel, & 1 batea.	2	..	10,417	..	174	Tailings are ground in 30 arrastras and concentrates chlorinated.
Nevada County— North Star	*g*	Do. ..	6 ft. Pelton, 4 ft. do., & engine.	2	30	850	6	12 Triumphs ..	1 clean-up, 1 barl., 1 batea.	..	..	....	..	..	.....
North Queensland— Charters Towers— Day Dawn Blk. (Burdekin) ..	*m*	Free gold in battery and plates and pans.	100 H.P. eng.	3	60	800	12	24 Frues, 18 Br. & Stansfield.	2 Berdans, 24 Wheelers.	12	..	52,146*n*	..	869	.....
Day Dawn P.C. (Excelsior)	*e*	Do. ..	132 H.P. eng.	1	40	896	..	8 Brn. & Stans.	38 Ber, 16 Wh.	8	..	50,000*o*	..	1,250	.....
Bonnie Dundee (Plants)	*g*	Do. ..	45 H.P. eng.	..	15	896	..	6 Brn. & Stans.	1 Ber., 11 Wh.	7	..	16,000*o*	..	1,066	.....
Defiance	*e*	Do. ..	40 H.P. eng.	..	15*q*	800	..	2 Buddles ..	2 Wh., 2½ Ber.	1	..	12,000*q*	..	800	With machinery since added, will probably come to about £1,100.
New Queen	*e*	Do. ..	68 H.P. en. with comp. cylrs. and var. expn. gear.	..	25	800	..	2 Buddles, 2 Br. & Stans., 5 end-blow tables.	39 Ber., 2 Wh..	3	..	20,000*o*	..	800	.....
Rishton— Disraeli	*m*	Do. .	50 H.P. engines	1	20	760	4	8 Frues ..	8 Ber., 4 Wh. *s*	2	..	20,000*ot*	1,753*u*	1,000	*v*; 4 132 × 50*j*.
Etheridge— Durham and Lord Byron	..	....	82 H.P. engines	1	20	900	..	2 Frues, 4 Br. & Stansfield.	10 Berdans, 8 Wheelers.	2	..	35,000*o*	..	1,750	Freight and labour are much more expensive than in the Charters Towers district.
Cumberland ..	..	....	92 H.P. engines	..	30	900	..	4 Frues, 4 Br. & Stansfield.	3 Berdans, 20 Wheelers.	3	..	35,000*o*	..	1,166	

TABLE I.—*Continued.*

a Laid out in steps with considerable fall, involving about 2,000 cubic yards of cutting in decomposed granite.
b As stated by Messrs. Fraser & Chalmers, who supplied the machinery.
c One pair of rolls, 20 inches × 13 inches.
d Frankfort mill.
e Slight fall.
f 7 Hungarian mills.
g Slightly graded.
h Batteries face to face in two rows, with tables in middle.
i Dimensions of mill building.
j Exclusive of engine-room.
k No. 5 Blake.
l Batteries back to back in two rows.
m Graded.
n Report of the Company, July 22nd, 1890.
o Report Department of Mines, Queensland Government, 1889.
p Grizzlies, boilers, etc.
q Report Department of Mines, 1879. (5 stamps, 3 settlers, 2 Brown & Stansfield concentrators, 2 percussion tables, 24 Berdans, and 2 Wheelers have been added since.)
r 1 Tangye double 10 inches cylinder geared pump, capable of pumping 35,000 gallons per hour.
s Large wooden-sided combination pans, 5 feet diameter.
t This sum includes machinery, grading, timber, and erection of building, fittings and hardware, and freight and duty; also a full stock of tools and supplies for twelve months.
u Published Report of Mr. C. P. Purinton, August 15th, 1887.
v Batteries in line.

"Charters Towers," and "Rishton" are within the Charters Towers gold-field.

The figures in the column of "Cost per Stamp" go to show that the more modern English mills, like the New Queen, Diarseli, and Day Dawn block, have been built cheaper than the mills locally erected previously in the same district. The Burdekin mill, which is an extremely fine piece of work, was built by a Maryborough (Queensland) engineer.

Hardwood boards, of different widths and thicknesses, cost about 23s. to 25s. per 100 B.M. in 1887-8. Kauri pine flooring, T. G. boards of ordinary width, cost about 30s. to 32s. per 100 B.M. in 1887-8. Cordwood costs 18s. to 22s. per cord.

According to the Queensland Government returns (from which the particulars given in Table I. are taken) another point to be noted is that the cost of some of the older Charters Towers batteries, like the Excelsior plant, seems to have been considerably in excess* of that of the newer mills, while even among these latter there are considerable variations of cost.

In this connexion the author would like to draw attention to several points which invariably affect the capital expenditure on any two different mills, and it is therefore of the first importance both to the mill-wright and the mill owner to make due allowance for them in any estimates made beforehand.

The prime cost of a mill, in fact, depends entirely on a variety of conditions:—

* This is no doubt partly owing to the higher cost of wages and freight in the early days of the field (everything having to be carted from Townsville on the coast), but, on the other hand, there was no duty to pay, and the price of bush timber was no doubt lower than it was five years ago.

(1) The size of the mill in regard to the tonnage of ore it is designed to handle.
(2) The general nature of the machinery and the mill site.
(3) The internal details of the building (in grouping and placing various machines together), and the choice of materials used in its construction.
(4) The distance the machinery and appliances composing the plant have to be transported from the place of manufacture, and the ease or difficulty of delivering them at the site of the works.
(5) The cost of the machinery at the foundry.
(6) The duties and commissions paid upon it.
(7) The local cost of labour and of timber, and other structural materials (purchased on the spot or elsewhere) delivered.
(8) The efficiency of the labour employed in their erection, and the time the works are under construction; which is contingent more or less on the willingness and ability of a company to provide for a heavy outlay covering a short period, or preferring payments distributed over a certain length of time.

To illustrate what is meant, the author will take two Queensland batteries with which he has been personally connected, the New Queen and the Disraeli.

The New Queen mill was designed and built on the best Australian plans, under the superintendence of Mr. Geo. Cavey; the erection of the machinery being entrusted to a first-rate English millwright and machinist, Mr. R. Robinson; and the battery has very justly earned the reputation of being one of the finest of its class on the Charters Towers field.

The Disraeli battery, on the other hand, was designed by Mr. J. Deby, of London, on American lines (the minor details and design of the building being left to the author's discretion), and it was erected under his supervision and that of his assistant, Mr. H. L. Lawrence; Mr. Wm. Reed, a most excellent English mechanic from the Sandycroft foundry (which supplied most of the machinery)* erecting everything in running order to plans under his orders.

Now, if we take the actual difference of cost per stamp in the New Queen mill and in the Disraeli, as shown by Table I., we find that the

* Messrs. Fraser & Chalmers furnished the Frue vanners and Messrs. Langland of Melbourne, the extra berdans.

latter plant (erected) apparently cost in proportion about £4,000 more than the New Queen battery, and it is a matter of interest to learn the cause of this discrepancy, of which the author can offer several explanations bearing on the points before mentioned.

Before referring to these, however, the author should point out that the figures given in Table I. as the cost of the Rishton mill are in reality fully £1,000 in excess of the actual sum expended, which should be £19,000, reducing the difference to £3,000. Further, the £19,000 in question includes the cost of a tram line, an expensive pumping station, and tailings-flume (the former to pump water from, and the latter to carry the tailings half-a-mile to, the river). These items do not enter into the cost of the Queen battery. Deducting their cost (according to figures given in a report by Mr. C. P. Purinton, published by the Company, Aug. 15th, 1887) we would have left (£19,000 — £1,605 18s. =) £17,394 2s., or an actual difference in cost of only £1,394 2s. between the two batteries. If, further, the cost of the reservoir in addition to the other outside works be subtracted, it would make the actual cost of the Rishton battery itself £17,246 13s. 6d., or £862 6s. 8d. per stamp, which is what doubtless it actually cost, viz., £8,146 spent on mill-building, construction, grading, and foundations, and the balance of £9,100 on machinery, freight, duty, etc. If a comparison were instituted on this basis it would make the difference in cost between the two mills only £62 6s. 8d. per stamp, or £1,246 13s. 6d. in total. We will assume, however, a supposed difference of £3,000 for the sake of illustrating a case in which this would be actually amply justified.

1. By the relative difference in the size of the two plants.
2. By the difference in the general nature of the machinery composing the two plants, and of their respective sites, which necessitated extra fittings and provisions, which could be dispensed with in the one case but not in the other. Such, for instance, as the cost of heavy framing and spacious bins,* required for the rock-breaker and automatic feeders of the Disraeli, and the extra strong framework needed for the combination-pans and large settlers, as compared with the small ordinary iron-wheelers, berdans, and settling-pans used at Charters Towers. To this may likewise be added

* In this connexion, it is a point worth noting that the planking of ore-bins are best laid on joists resting across the inclined timbers of the framework. The sheathing will then run lengthways down the bin and not across it; if double planked, the joints should be "broken" by covering the lower ones with the top boards.

the extra space taken up in the Rishton mill by the vanners,* which require a solidly and evenly boarded floor, and the fact that in a mill like the Disraeli, provision must of necessity be made to protect the superstructure as far as possible from the attacks of white ants, by bedding the mud-sills of the building on masonry, which is not such a necessary object where a rough stick can be put in or taken out without any great difficulty.

3. By the arrangement of the machinery and consequent difference of design of the internal details of the buildings. The Disraeli battery, being built in steps and storeys, requiring heavy grading, faced with retaining walls of suitable strength, and a massive framing of squared-timber† to support the load of the building and machinery, so as to provide sufficient fall for the ore to descend by gravity, with as little handling as possible, through the mill.

   The New Queen, on the other hand, like most of the other Charters Towers batteries (except the Burdekin and Plants), possessing such slight fall as to allow of the use of ordinary round timber (barked, but otherwise untrimmed) in the construction of the buildings, and requiring no expensive foundations, except for the mortar-blocks of the stamps themselves.

4. By extra freight; Rishton, where the Disraeli battery was located, being 22 miles from the railway terminus (which is at Charters Towers itself), so that an additional charge of £2 to £2 5s. a ton can be reckoned on all material used, representing a by no means insignificant item in the sum total.

Table II. gives the cost of treatment in a tabulated form at several of the mills, which have been referred to in Table I., and will be found useful for purposes of comparison.

* The cost of adding 24 vanners to the Montana Company's mill, at Marysville, is stated to have been £3,437 10s., shed and machinery included.

† Imported Oregon pine; as sticks of sufficient size and length (some running up to 12 inches by 12 inches by 41 feet) could not be obtained in the district.

TABLE II.

Showing the Cost of Free-milling and Grinding-milling Treatment in various Localities.

	Europe.			Asia.		South America.	Australia.						New Zealand.	South Africa.	
	Wales.		Italy.	Wynaad.	Mysore.	Venezuela.	Burdekin	Charters Towers.			Rishton.	Victoria.	The Thames.	Johannesberg.	
Name of mill	40-Stamp Mill. New Morgan ..		Pestarena. Cost per ton of picking and crushing 1t. stone breaker and rolls, 2s. 8½d. Do. of amalgamation, including mercury, wear and tear, etc., 6s. 2d.	20 Stamp. Indian Phœnix.	90 Stamp Mill.	60 Stamp. El Callao	60 Stamp. Day Dawn Block.	40 Stp. Day Dawn	50 Stp. P.C.	25 Stamp. New Queen.	20 Stamp. Disraeli.	60 Stamp Mill. S. Clunes United.	33 Stamp Mill. Saxon.	20 Stamp. May Consolidated.	100 Stmp. The Jumpers.
	1891.														
Tons crushed	960 (= 21,760 12 mos.)	1,700 (= 19,550 12 mos.)		1884. 250 (= 12,574 12 mos.)	1892. 38,727	1891. 58,949	1890-91. 20,078	1889. 21,920	1890. 27,636	1889. 5,113 (= 10,226 12 mos.)	1887. 890 (= 10,680 12 mos.)	? 28,820	? 63	1889-90. 14,992	1889-90. *w* 24,795
Time crushing lasted ..	320 hours	627½ hrs.		6 days.	12 mos.	12 mos.	12 mos.	12 mos.	12 mos.	6 months	1 month	12 mos.	24 hours.	12 mos.	6 mos.
Labour per ton crushed, cost	s. d. 0 8	s. d. 0 10½		s. d. 0 11⅙	s. d. ..	s. d. ..	s. d. 4 3½	s. d. 6 5	s. d. 5 5½	s. d. 5 10	s. d. 4 9½	s. d. 0 10½	s. d. 2 10½	s. d.	s. d.
Material	0 2	0 6¼		1 0½	..	..	0 11	2 0½	1 6¼	1 5	1 0				
Fuel	..			Nil.	..	..	1 7½*m*	2 4*p*	2 3*p*	2 2½*p*	1 3½*m*	1 4½	?	13 0½	10 1
Repairs	?	?		0 8½	..	..	1 1½	0 6¾	0 1	0 4½	0 1½		1 2½		
Transport	0 6½	0 7		0 5	..	..	2 2*n*	0 6½*q*	0 3½*r*	2 4*s*	0 6*t*	0 8¶	?	2 6¾	0 10½
Water	Nil.	Nil.		Nil.	..	..	Nil.	Nil.	Nil.	Nil.	1 5½*u*	Nil.	Nil.	Nil.	Nil.
Total cost per ton, using automotive power — Water ..	*a*1 4½	..	*b*8 10¾	*c*3 0¾	..	..	..	..	..	..	..	..	..	..	..
Steam ..	..	..	..	..	†14 6	*d*5 0	*f*10 1½	*g*11 10¾	*h*9 7¾	*i*12 2	*j*9 2	2 11	4 1	*v*15 7½	*x*10 11½
Water and steam ..	..	*a*1 11½	..	..	..	..	..	..	..	..	..	..	..	..	..

TABLE II.– *Continued.*

	North America.						
	Alaska.	Dakota.				Colorado.	California.
Name of mill	240 Stp. Alaska-Treadwell.	80 Stamp Mill. Homestake ..		120 Stamp Mill. Golden Star.		75 Stamp Mill. Hidden Treasure.	40 Stamp Mill. Empire.
Tons crushed	1890-91. 220,686	1887-88. 96,790	1880 53,372	1880. 121,910	1887-88. 146,565	1891. 30,720	? 21,000
Time crushing lasted ..	12 mos.	12 mos.	12 mos.	12 mos.	12 mos.	12 mos.	12 mos.
Labour per ton crushed, cost	s. d. 0 9½ *b*	s. d. 1 0½	s. d. 2 2½	s. d. 1 4½	s. d. 0 10½	s. d. 1 7	s. d. 0 10½
Material	0 6½ *b*	*0 2½	0 8½	0 3½	0 2½		0 5½
Fuel	0 3½	1 1½	1 2½	1 0½	1 1½		
Repairs	0 1½	0 4½	0 8	0 1½	0 6½	1 8	0 4
Transport	?	?	0 1½	?	?	?	?
Water	Nil.	0 8½	0 5½	0 5½	0 8½	?	Variable.
Total cost per ton, using automotive power — Water ..	*c* 1 9	..	..	..	..	..	1 8‖
Total cost per ton, using automotive power — Steam ..	..	*g* 3 5½	*z* 3 4½	*z* 3 2½	*g* 3 5½	..	..
Total cost per ton, using automotive power — Water and steam ..	..	..	..	..	..	3 3½	..

a Manager's Reports, June to November, 1891.

b Estimate of "the agents."

c Report by the writer, August, 1884, published by the company, January, 1885. With a 20-stamp mill, possessing good fall, arranged with rock-breakers, stamps (with inside and outside "coppers"), and vanners, these costs could probably be reduced to 1s. 10d. per ton in India under like conditions.

d Report of Mr. George E. Webber, Jun., Superintendent, 1891. (6·29 fr. = 5s. per ton.) For the year 1891, 1,589 cords of wood were used, equivalent to 800 tons of coal of fair quality, costing £4 7s. per ton. In addition to crushing 58,942 tons of quartz, the engine did other work (pumping water and driving dynamo for hoisting-works, etc.), so only about three-quarters of the power employed was used for milling. Hence fuel used per ton of ore was equivalent to about 22 lbs. of good coal. The engine used at El Callao is a compound condensing tandem, supplied with steam by locomotive boilers with combustion chambers; steam pressure, 140 to 150 lbs. Mr. Hamilton Smith, Jun., (Notes on Gold Quartz Milling at El Callao, March, 1892) remarks, considering the high cost of labour supplies and fuel at El Callao: – "This is an exceedingly low rate. The above cost of 6·29 fr. per ton includes 0·64 fr. per share of general expenses. Such a charge is not generally made to milling accounts, so on the usual basis, the cost per ton for milling was 5·39 fr. or 4s. 6d."

e Company's Annual Statement, 1891.

f Company's Report, July 14th, 1891.

g Company's Report, 1889.

h Company's Report, 1890.

i Manager's Report, December 31st, 1889.

j Estimated by the writer. Mr. C. P. Purinton, who reported on the Disraeli in 1887, stated in his Report published by the Company, page 9, that "the cost of milling depends in a great measure on the quantity milled," adding:– "If the mine supplied ore enough in sufficient quantity to keep the mill constantly employed, the ore could be milled for not to exceed 5s. per ton, whereas the cost now is fully 10s." If general expenses and management (which for the sake of uniformity with the other mills is not included in the author's figures) be apportioned and added to his estimate, it tallies almost exactly with Mr. Purinton's. An extra charge per ton milled ought to be added to all estimates for the amortization of the capital, calculating it at from 10 to 15 per cent. of the gross cost of the plant, according to the probable life of the works. It is not unusual to write off 5 to 10 per cent. on machinery and 2½ to 5 per cent. on buildings; but this proportion depends on circumstances, where (as in some cases) the materials of the building when "sold off" would command a better price comparatively speaking than the machinery.

TABLE II.—*Continued.*

ALASKA-TREADWELL.

k Subdivision of Labour—	Cost per Ton. Dol.
Foremen	0·0215
Amalgamators	0·0258
Feeders	0·0402
Oilers	0·0098
Concentrators	0·0333
Rock-breakers	0·0168
Do. (whites)	0·0343
Ditchmen	0·0090
Do. repairs	0·0033
Total	0·1940

l Subdivision of Material—	Cost per Ton Dol.
Shoes and dies	0·0606
Concentrator fittings	0·0078
Screens	0·0034
Rock-breaker supplies	0·0059
Feeder do.	0·0032
Miscellaneous	0·0032
Guide-blocks	0·0009
Oils and lubricants	0·0013
Lumber	0·0015
Rope and hose	0·0029
Water-wheel supplies	0·0019
Mortar and aprons	0·0062
Mercury	0·0088
Cam-shafts	0·0022
Battery linings	0·0029
Electric light	0·0062
Total	0·1211

m The Disraeli and, the writer believes, the Day Dawn Block use tubular boilers.

n Transport by rail (12 miles).

o In 1889, 10 stamps, 12 Berdans, and 2 concentrators were added to the plant (specified in Table I.) and came into operation the following year.

p The Day Dawn P.C. and New Queen use Cornish boilers, owing to the corroding action of the Charters Towers water. Fuel is also somewhat cheaper at Rishton and the Burdekin than in the Towers.

q Transport by rail (1½ miles). *r* Ditto (1½ miles). *s* Transport by cart (¾ to 6 miles).

t Transport by horse-tramway (½ mile).

u Labour, 10½d.; fuel, 6½d. Pumping 150 gallons 2,500 feet, with a vertical lift of 106 feet.

v Report of the Company for year ending May 31st, 1890.

w Average duty of stamps, 2 tons 14 cwts. 3 qrs. per 24 hours.

x Report of the Directors for the half-year ending January 31st, 1890. The cost of milling during this period varied, it may be remarked, from month to month (from as low as 9s. 2d. per ton crushing 4,926 tons; to 22s. 8¾d. crushing only 1,245 tons).

y "Gold Milling in the Black Hills," by H. O. Hofman.—*Trans. Am. Inst. Min. Eng.*, vol. xvii., p. 498.

z Tenth Census of the United States, page 280.

* Includes supplies, candles, oil, mercury, lumber, and timber.

‡ Water power used for four months, steam for four months, and both water and steam for remainder of year.

‖ This figure is an average, the cost varying from 1s. 5½d. to 1s. 10½d. When steam is used 5d. extra must be added to the total given.

¶ Tramming the stone one-third mile and breaking it on contract.

† Of this sum 5s. to 6s. 6d. is probably chargeable to treatment of tailings by grinding.

NOTE.—To institute a fair comparison between the total cost at the different mills recorded in the table, the cost of transport must, of course, be deducted from the total cost, making allowance as well for difference in the relative quantity crushed in each case, etc.

The cost of treatment in different localities, using the same milling process, is quite as variable as the cost of plant.

The author does not think he is overstating the fact in saying that at most of the older Charters Towers batteries (with 15 to 20 heads) the cost of milling runs from 14s. to 18s. per ton and more, including only such items as are given in Table II.

At Plant's mill, according to the report of the company (*Northern Miner*, September 19th, 1889), the profit for the year ending December 31st, 1888, was about £3,000, and if the stone milled be taken at 13,500 tons and the charge for crushing, including grinding, is assumed to have

TABLE III.

SHOWING RULING RATES OF WAGES IN DIFFERENT LOCALITIES AND STAFF OF VARIOUS MILLS.

Occupation.	20 Stamp Mill, India.			80 Stamp Mill, Dakota.*			120 Stamp Mill, Dakota.†			40 Stamp Mill, Wales.‡	
	No. of Men.	Length of Shift.	Wages per Shift.	No. of Men.	Length of Shift.	Wages per Shift.	No. of Men.	Length of Shift.	Wages per Shift.	Length of Shift.	Wages per Shift.
		Hrs.	s. d.		Hrs.	s. d.		Hrs.	s. d.	Hrs.	s. d.
Foremen	1	...	...	¾	...	27 1	¾	...	27 1	...	...
Amalgamators ...	2	12	3 4	1	10	16 8	1	10	16 8	12	6 6
Assistant do. ...	1	12	1 5½	4	12	14 7	4	12	14 7	12	5 6
Rock-breakermen	...	...	...	5	10	12 6	6	10	12 6	12	4 0
Feeders & stone-breakers ...	14	12	0 8¼	2	12	12 6	4	12	12 6	12	4 0
Oilers	1	12	0 8¼	2	12	12 6	2	12	12 6	...	...
Machinists ...	..	...	3 4	1	10	17 8½	1	10	17 8½	...	...
Pipe-fitters ...	...	...	...	½	10	14 7	½	10	14 7	12	4 6
Engine-drivers ...	...	...	3 5½	2	12	14 7	2	12	14 7	...	...
Firemen ...	...	...	...	2	12	12 6	2	12	12 6	...	...
Blacksmiths ...	1	12	1 8	...	...	...	...	...	...	...	...
Strikers	1	12	0 7¾	...	...	...	...	...	...	...	...
Carpenters ...	1	12	1 8	...	...	...	...	...	...	...	...
Labourers ...	2	12	0 8¼	...	...	...	...	...	...	8	3 6
Watchmen, etc....	4	12	0 8¼	½	12	12 6	½	12	12 6	...	...
Panmen & oilers	...	...	...	...	...	...	...	...	...	...	...
Vanner-attendants	...	...	...	...	...	...	...	...	...	...	...
Total number	28	...	...	20½	...	...	23½	...	...	...	...

* Homestake. † Golden Star. ‡ Morgan G.M.C.

been somewhere about the usual price of 18s. to 22s. per ton,* then taking it at 19s. per ton, it leaves a balance of £9,825 for milling expenses during the year, which, divided by the gross tonnage crushed, would represent a milling cost of about 14s. 6¾d. per ton. This is a case of special interest, because wheelers are used entirely for grinding and amalgamating, to the exclusion of berdans, in which respect this mill differs from most of the others in the district. An examination of Table II. shows, however, that the cost of milling in the same locality varies greatly, depending upon:—

1. The general design and internal arrangements of the mill-building, for economizing and facilitating labour, and simplifying the plant, depending to a great extent on the selection of a suitable site and its proper utilization.
2. The general nature of the process in regard to the saving or loss of gold and mineral, which the disposition and character of the machinery effects.

* Depending on the amount of pyrites in the stone, and the time taken to grind a given tonnage (varying with the capacity of the pan).

TABLE III.—*Continued.*

Occupation.	20 Stamp Mill, Disraeli, Rishton.			Charter's Towers.					
	No. of Men.	Length of Shift.	Wages per Shift.	A 10 Stamp-battery	A 15 Stamp-battery	A 20 Stamp Mill.	A 25 Stamp Mill.	Length of Shift.	Wages per Shift.
				No. of Men.	No. of Men.	No. of Men.	No. of Men.	Hrs.	s. d.
Foremen	...	...	Rates much the same as at Charters Towers.	...	..	...	...	...	...
Amalgamators ...	1	12		2	2	2	2	12	16 8
Assistant do. ...	2	8		...	...	...	...	8	13 4
Rock-breakermen	1	8		...	...	...	...	8	10 0
Feeders & stone-breakers ...	1	8		3	4	5	6	8	10 0
Oilers	...	...		...	...	...	...	...	...
Machinists ...	½	...		1	1	1	1	12	16 8 18 4
Pipe-fitters ...	...	...		...	...	...	...	...	...
Engine-drivers ...	3	8		3	3	3	3	8 12	11 8 13 4
Firemen ...	...	...		...	...	...	...	...	...
Blacksmiths ...	...	...		...	...	...	...	8	13 4 16 8
Strikers	...	...		...	...	...	...	8	10 0 11 8
Carpenters ...	...	...		...	...	...	...	8	15 0
Labourers ...	...	...		...	...	...	...	8	10 0
Watchmen, etc....	...	...		...	...	...	...	8 12	10 0 11 8
Panmen & oilers	2	...		1 to 2	2	2	3	8	10 0 11 8
Tailingsmen and Vanner attend'ts.	1	...		*1	*1	*2	*2	12	11 8
Total number	11½	...	...	11 to 12	13	15	17	...	...

* Tailingsmen.

3. The gross tonnage that is handled by the mill in a given time (sometimes affected by climate).
4. The efficiency of the labour employed, and its cost.
5. The quality and the price of supplies, fuel, etc., used.
6. The power employed, and its method of application.
7. The situation of the works as regards water-supply, transport of ore from the mine to the mill, and disposal of tailings.
8. The efficiency of the general management.

All these points have to be studied from the standpoint of relative utility and comparative cost.

As regards the author's first proposition, effect of general design, nothing could illustrate what is meant better, than the saving of labour and material (as shown by Table II.) effected in mills, laid out like the Day Dawn block and Disraeli, as compared with batteries of the New Queen and Day Dawn P.C. type.

TABLE III.—*Continued.*

Occupation.	60 Stamp Mill, Dakota, California.			40 Stamp Mill, California Empire.			75 Stamp Mill, Colorado, Hidden Treasure.			35 Stamp Mill, New Zealand, Saxon.		
	No. of Men.	Length of Shift.	Wages per Shift.	No. of Men.	Length of Shift.	Wages per Shift.	No. of Men.	Length of Shift.	Wages per Shift.	No. of Men.	Length of Shift.	Wages per Shift.
		Hrs.	s. d.		Hrs.	s. d.		Hrs.	s. d.		Hrs.	s. d.
Foremen... ...	1	...	27 1	...	...	...	...	...	...	...	...	...
Amalgamators ...	2	12	15 7½	2	12	12 6	1	12	24 3½	3	8	8 0
Assistant do. ...	...	...	...	...	...	...	1	12	13 10½	...	...	...
Rock-breakermen	1	10	12 6	1	12	10 5	...	...	...	...	...	...
Feeders & stone-breakers ...	2	12	13 6½	...	...	...	6	12	12 6	3*	8	6 8
										3†	8	5 0
Oilers	...	...	...	...	...	...	...	...	...	...	...	...
Machinists ...	1	12	18 9	...	...	...	...	...	...	...	...	...
Pipe-fitters ...	...	...	...	...	...	...	...	...	...	...	...	...
Engine-drivers ...	2	12	14 7	...	...	...	...	...	...	...	...	...
Firemen	2	12	13 6½	...	...	...	...	...	...	...	...	...
Blacksmiths ...	...	...	...	...	...	...	...	...	...	...	...	...
Strikers	...	...	...	...	...	...	...	...	...	...	...	...
Carpenters ...	...	...	...	...	...	...	...	...	...	...	...	...
Labourers ...	1	10	10 5	2	12	10 5	...	...	...	...	...	...
Watchmen, etc....	...	...	...	...	...	...	...	...	...	...	...	...
Panmen & oilers	...	...	...	...	...	...	...	...	...	...	...	...
Tailingsmen and vanner-attendants	...	...	...	1	12	12 6	2	12	12 6	3	8	3 4
Total Number	12	...	...	6	...	...	10	...	...	12	...	...

* Men. † Boys.

Wages in Wales run from 21s. to 30s. per week. The staff is composed of 7 men by day and 3 by night at the Morgan mill.

Wages in Italy run from 9½d. to 3s. 8d. per day at Pestarena. Girls earn 9½d. to 11d. per day; millmen, 2s. per day; labourers, 1s. 6½d. to 1s. 9d. per day; Smiths, 2s. 5½d. to 3s. 8d. per day.

Wages in South Africa are as follows:—Amalgamators (European), £20; assistant amalgamators, £15; rock-breakermen, £17 10s.; machinists, £30; engine-drivers, £26; blacksmiths, £30; carpenters, £22 10s.; and labourers, £13 per month. Natives earn £3 per month. The Jumpers mill employs a staff of 13 Europeans and 65 Kaffirs, with 4 European vanner-attendants and 5 Kaffirs in the vanner-house.

As both the Day Dawn batteries are first-rate mills of their class, and are in charge of first-class Queensland mill-men, and both the New Queen and Disraeli were under the writer's management, when the figures given in Table II. were compiled, he has no hesitation in taking these four plants for comparison. In regard to the New Queen, it will be noticed that the charge for labour, material, and repairs amounted to 7s. 7½d. per ton in 1889, as compared with 5s. 11d. at the Disraeli in 1887, a difference in favour of the latter of 1s. 8½d. per ton, notwithstanding the extra charges on supplies, etc., involved in transporting goods of every sort, an additional 22 miles to Rishton. What lends these two cases special value is the fact that, for all practical purposes, the 3rd, 4th, 6th, and 8th of the above considerations may be considered practically the same in both. The 1st, 2nd, 5th, and 7th are therefore evidently what really affect the point under consideration.

The nature of the plant affecting as it does the cost of treatment, must be always recognised in considering the cost of a mill. Now, if we assume, for instance, an equal tonnage crushed in the New Queen and Disraeli batteries, amounting to 10,000 to 12,000 tons per annum (running the same number of stamps) it is evident that there is a gross saving of between at least (1s. 8½d. × 10,000 =) £854 3s. 4d. to (1s. 8½d. × 12,000 =) £1,025 annually in favour of the Rishton works. Against this, strictly speaking, however, one must place the interest (reckoned say at 4 per cent.) on the extra capital outlay expended on the latter mill which, if we include accessory plant (such as reservoir pumping-station and race) is assumed to amount as before stated to about £3,000.

It will be noted that if one were to take the Burdekin or Excelsior batteries for comparison, from the costs given in Table I., the difference in the former case on 20 stamps erected would be £2,620 against the Disraeli. This, however, is easily explained by the difference in freight rates and the proportionately lower cost of constructing a large mill as compared with a small one; and as a great deal of the machinery was got from Maryboro' it had not to pay duty. In the case of the Excelsior mill it would be (20 × £250 =) £5,000 in favour of the Rishton mill.

It is indeed more than probable that, given the same mill-site, if the Disraeli battery had been erected at Charters Towers, it would have cost but little more than the New Queen per stamp-head, but taking the assumed capital expenditure in the two cases, and deducting interest as before remarked on £3,000 at 4 per cent., or £120; on the most unfavourable supposition, we might fairly attribute a net economy per annum of (£854 3s. 4d. — £120 =) £734 to (£1,025 — £120 =) £905 to be credited simply to the general design of the building and character and disposition of the machinery, which would repay the assumed extra capital cost with interest in between 3 and 4 years. What applies in this case evidently applies equally to any other similar 20 stamp mill in Queensland, entirely apart from the extra saving of gold a properly arranged mill is likely to effect; a matter which, though intimately connected with the question, comes under the author's second heading.

The saving that may be effected by arrangement of plant would, however, be far more striking if one had for comparison a 40 stamp mill, built like the Disraeli with a 40 stamp mill of the ordinary Charters Towers type, since the former could be run with 4 or 5 extra

men making say 16 all told, whilst the latter would require with the most careful management at least 24; while for an 80 stamp mill the numbers would stand relatively about as 26 to 44.

Reckoning mill wages on the average at 10s. 7d. per day (shift) and the working year at 303 days, this represents a saving on wages alone, running a 40 head battery, of £4 4s. 8d. per diem or £1,282 14s. per annum, or on an 80 stamp mill £2,886 1s. 6d. per annum, *plus* the extra saving on supplies, repairs, etc., which reckoned at only 8d. per ton would amount to a considerable sum, say £666 to £1,333 additional.*

The first cost of machinery is often only a small part of the total cost of erection, and it is therefore of the first importance that the best design and execution should be insisted on.†

Again, Messrs. McDermott and Duffield cite the case of two gold mills in Venezuela, running side by side, and owned by the same company, which well illustrates this. Both are 66 stamp mills operated on the same ore. The one poorly designed and built, the other embodying the results of practical mill-men's experience. The one mill can be made to average 93 tons crushed daily. The other averages over 143 tons. The first costs in working expenses, 18s. 9d. per ton of ore, the latter only 6s. 3d. These mills are ordinary gold mills, and it will appear incredible to those who have not run such machinery that such differences can exist, believing that a stamp mill is merely a medium for crushing, and that economy always consists in purchasing it in the cheapest market and erecting it anyhow.

The author's second proposition, viz., the saving or loss of gold, the details of the arrangement of a mill effects is one that is exceedingly complex, as there is an infinite choice of machinery and method of

* As bearing on this question the report of the Superintendent of the El Callao, 1891, may be quoted:—"Notwithstanding the increased amount of work the mill has had to perform and its growing age, few renewals and repairs have been required, and at the close of the year all its parts remain in good running condition."

† Mr. Hamilton Smith, Jun., says: "The old El Callao mill for the year 1882 crushed 22,405 tons of quartz at a cost of 78·30 francs per ton. A comparison of these results with the results (see Table II.) obtained by the new mill is very instructive to mining men; as there could be no better proof of the advantages of first class mining machinery coupled with judicous management. Had it not been for the construction of this new mill, El Callao mine would have suspended operations years ago." The cost for the year ending December 31st, 1892, was 6·22 francs per ton, whilst the average assay of the tailings was 3½ dwts. per ton.

grouping it, which more or less affect the question, depending on the nature of the ore; various illustrations of this will, however, be given later on.

It is a matter which must be confided to the technical knowledge and experience of the engineer who designs the plant presumably with certain specified objects in view, a number of which have been incidentally noticed. The losses in gold-milling are, like local costs, extremely variable.

In Gilpin County, Colorado, for example, 113,427 tons were milled during the census year ending May 31st, 1880, out of which :—

Per Cent.			
65	of the gold was saved by	direct amalgamation.	
4	„	„	pan treatment
7·05	„	„	re-concentration of tailings.
Total, 76·05	The ore running £2 1s. 3½d. in gold and 1s. 3d. in silver.		

The above agrees with the writer's experience in India, where he found 76·2 per cent. was saved by direct amalgamation and pan treatment.

The latest recorded results of one of the best mills of Gilpin County, which is said to be fairly representative of present Colorado practice, shows that treating an ore containing 7·46 ozs. of gold and 32·86 ozs. of silver, it is possible to extract by milling 93·8 per cent. of the gold and 74 per cent. of the silver (including the value in the concentrates) out of which 70·4 per cent. of the gold and 42·6 per cent. of the silver was extracted by direct amalgamation. This, the writer thinks, shows, that whilst the saving by direct amalgamation in Colorado remains much the same as it was, a steady improvement has taken place in the manipulation of the pyrites.

Treating 25 dwts. ore at El Callao the loss is said to be 3½ dwts.; in 1891 it was 2¾ dwts. In Dakota, the Homestake and Golden Star mills claim to save 85 per cent. of the free gold, ½ to ¾ dwt. being unrecovered in the tailings. The tailings of the Alaska Gold-mining Company run ½ to 1 dwt.

Mr. G. T. Deetken, some years ago, made some elaborate experiments at one of the best Californian gold mills of the day, and found that 27 per cent.* was lost in the tailings giving an extraction of 73 per cent. At the present time, with improved methods of treatment, the extraction

* This, again, about corresponds, the author believes, with the average loss (gross) in the Charters Towers mill tailings, crushing and treating 25 dwts. ore; it may be assumed that in concentrating and grinding the pyrites, the loss runs from 5 to 7 dwts.

in California probably approaches nearer 80 to 85 per cent., and one instance might be cited where it was as high as 82 to 94 per cent., the concentrates being treated by chlorination.

As most gold ores contain some gold combined with pyrites, in all probability, the average extraction of most mills treating stone of this kind often falls short of 75 per cent., and is occasionally as low as 45 per cent. if the stone contains much pyrites, which is dealt with in a crude manner.

In Victoria, treating $5\frac{1}{4}$ dwts. to $5\frac{3}{4}$ dwts. ore, the average extraction at the present day in the best mills is about 84 per cent., and has run as high as 87·6 per cent., which is the best record grinding milling can show under the most favourable circumstances with an ore exceptionally free. The milling practice of the district has steadily improved since 1861, when there was actually a loss of nearly 50 per cent. of the gold in the stone.

Though miners, as a class, possess just as much honesty and hard common-sense as people who follow many other callings, when it comes to consideration of these mill losses (although they can only be remedied by knowing exactly where the loss occurs and what it amounts to) the behaviour of a certain section of mining people and mill-men when confronted with facts of this sort reminds one forcibly of the behaviour of the ostrich confronted with death. The bird, it is said, sticks his head into the desert sands, and affects to disbelieve in the possibility of loss of life. The unpractical individual who generally loves to self-style himself the practical man* (who don't believe in assays, underground-surveys, and that kind of thing!) buries his eyes in the mill-sands, declares, believing it or not, that there is no loss of gold, and the facts, whether, as in some cases, from honest conviction, in others from dishonest expediency, become falsified.

It is curious that shareholders in mines seem actually to prefer to be deceived in this respect, and the mill-man who tells them that there is a loss in their tailings is lucky if he escapes being abused as well as thought a fool, simply because his neighbours choose to declare that they are losing nothing!

In this connexion, automatic samplers may be used with advantage to sample the crushed ore, pulp, and tailings regularly.

In August, 1884, the author put through a test crushing (in India) of

* A very different individual to the genuine miner or mill-man, of whom the late Mr. George Langtry, of Comstock fame, might be taken as a real type. See "Amalgamation on the Comstock Lode, Nevada," by Mr. A. D. Hodges, *Trans. Am. Inst. Min. Eng.*, vol. xix., page 216.

280 tons of stone, the particulars of which (given below) may be of some interest. It may serve as an illustration of the kind of information which ought to be obtainable of the working of every gold mill, and shows how, if the ore is regularly assayed (as it ought to be constantly), the results can be checked.

Number of stamps, 20; weight of stamps, 850 lbs.; lift, 9 inches; speed, 58 drops per minute.
Duty of battery, 41·3 tons per diem, or 2 tons 145 lbs. per stamp.
Order of drop, 3, 1, and 5 together, and 2 and 4 together.
Water supplied to each battery of 5 stamps, 3·52 cubic feet, or 22 gallons per minute.
Water supplied to other parts of the mill, 16 gallons per minute.
Ore per cubic foot of water discharged from the battery, $\frac{1}{3\cdot6}$ cubic feet.
Mesh of screens, 196 holes per square inch (about $\frac{1}{3\cdot7}$ inch).
Wear of shoes, $1\frac{1}{2\cdot5}$ lbs. per ton of ore crushed.
Wear of dies, about half that of the shoes.
Amount of pyrites in the ore, $2\frac{1}{2}$ per cent.
Degree of concentration, 50 per cent.
Concentrates obtained from percussion-tables, 14 tons.
Yield of milled gold, 2 dwts. 10·7 grains.
Bullion fineness, 911·5
Tons of ore crushed, 280.
Time crushing lasted, 162 hours 35 minutes.
Mercury required to charge entire mill, 1,458 lbs.
Loss of mercury (owing to an accident at one of the mills this cannot be stated exactly).

	By Assay.	By Estimate.	Actually Obtained or Accounted for.
	Ozs. dwts. grs.	Ozs. dwts. grs.	Ozs. dwts. grs.
280 tons of ore, assaying 3 dwts. 5 grs. per ton contain	44 18 8	...	...
Gold obtained from stamp-boxes	...	...	2 12 0
Gold in the ore leaving the battery (deducting gold obtained from boxes)	...	42 6 8	...
Gold obtained from copper-plates	...	...	18 10 0
Gold in the ore after passing the plates (deducting gold obtained from plates and boxes)	...	23 16 8	...
Gold obtained from riffles	...	...	4 0 0
Gold in the ore after passing the riffles ...	*a*18 14 12	19 16 8	...
Gold obtained from the wheeler-pans ...	...	...	6 10 0*b*
Gold in the concentrated slimes leaving the wheeler-pans	12 5 0	12 14 12	...
Gold obtained from the Hungarian mills ...	...	...	2 13 0*b*
Gold in 9½ tons of settlings, collected in catch-pits	7 8 1	...	7 8 1*b*
Gold in the tailings of the percussion-tables	1 12 0	1 11 20	1 11 20*b*
Gold in the tailings of the settling-tanks, and unaccounted for	...	1 13 11	1 13 11*b*
			44 18 8

The sum of the results marked *b* (19 ozs. 16 dwts. 8 grains) should equal the assay marked *a* (18 ozs. 14 dwts. 12 grains) to be theoretically absolutely accurate.

No efficient check can be kept on the work going on at a mine or mill without a proper record of the working of each department being kept by its immediate heads, and a subdivision of accounts, which shows the cost per ton, under different general heads, in detail from month to month.

The author's third proposition (the influence on cost of the gross-tonnage handled in a mill in a given time) is instanced by the results of milling at the Day Dawn P.C. mill in 1890 as compared with 1889, the effect of milling 1,140 tons extra in the latter year being to reduce the expense on each item of cost (Table II.).

Referring to the capital charges in 1890, the directors in their report make the following remarks:—"The capital expenditure in question has been fully justified, for it has already greatly reduced the working charges, as will be seen from the following table:—

Year.	No. of Tons Treated, excluding Mundic Stone.*	Total Cost of Reduction and Extraction of Gold.†	Cost per Ton of Ore Treated.‡
		£ s. d.	£ s. d.
1889	26,551	45,896 12 10	1 14 7
1890	28,879	42,924 16 7	1 9 9
	+ 2,328	− 2,971 16 3	− 0 4 10

showing that although 2,328 more tons were treated in 1890 than in 1889 the cost was £2,971 16s. 3d. less." This statement should be obviously reversed; the capital expended having increased the output in 1890 being the cause of the reduced cost.

Take once more, for example, the New Queen. The author has shown in Table II. that the average cost of treatment in 1889 was 12s. 2d. per ton, but as was remarked in the report published from which these figures were taken—from September to November (of the year in

* Including ore crushed at outside mills (the amount crushed at the company's own mill being given in Table II.).

† Including mine expenditure, mill expenditure, expenditure on Rose of England lease, crushing ore at outside mills, and bullion-remittance expenses account.

‡ The cost, including general expenses at Charters Towers, appears to have been £1 16s. 1¼d. in 1889 and £1 10s. 9¼d. in 1890.

question) the actual cost ranged from 9s. to 10s. 4d. per ton, according to the quantity of stone crushed, which varied from 1,162 tons to 943 tons per month.

The effect of the quantity milled on the cost of milling is again evident if we compare the cost of milling at the Homestake and Golden Star mines in 1880 (Table II.), and the cost of milling in the new 60 stamp mill* of the El Callao Company, during several different years, tabulated below ;—

Year.	Tons Crushed.	Cost per Ton. Francs.
1888	15,692	18·40
1889	43,629	12·60
1890	53,977	7·82
1891	58,949	6·29

The reason for this reduction is evident: with an enlarged scale of treatment, while the standing charges for management, skilled labour, etc., remain much the same, the amount of even the extra manual labour required is not by any means always increased in direct proportion to the additional tonnage crushed. It is to be recollected, however, that in crushing an extra quantity of stone its general average yield is likely to drop, as it often pays to crush stone in a large mill, which, with a smaller plant, would have to be discarded.

The yield of the Jumpers Company, for instance, crushing for six months with 30 stamps, fell from 18½ dwts., to between 10¾ and 11¾ dwts., with 70 stamps; and adding 30 stamps to the Robinson 10 stamp mill, the average grade of the ore fell 50 per cent.†

The Charters Towers gold-field, again up to 1889, is reputed to have produced 1,915,051 ozs. of gold representing a yield of 1 oz. 10 dwts. 9 grains per ton ; in 1890, 121,406 tons 8 cwts. 2 qrs. of quartz were crushed for an average yield of 1 oz. 6 dwts. 19 grains; and 174,000 tons in 1891, which only averaged 1 oz. 5 dwts. 9 grains.

Allowance must consequently be made for this decreased yield per ton in estimating the probable profits of a large plant when calculations are made, based upon the yield of an existing small mill.

Another point, however, in favour of large mills is that labour can be more easily specialized, which tends to reduce cost by promoting increased manual dexterity in individual operations, whilst by running reduction works up to their full capacity (a thing to be always aimed at) the fractional excess of unproductive labour, or, in other words, the time

* Supplied in 1885 by Messrs. Fraser and Chalmers.

† Messrs. McDermott and Duffield, page 5.

thrown away that has to be paid for, to perform duties which do not occupy one or more men a full shift, is reduced to a minimum.

The author's fourth proposition (the influence of the efficiency of the labour employed on the cost of milling) has a most important bearing on the question. It has been admirably dealt with by Mr. W. M. Howe in "Notes on the Bessemer Process."* Though speaking of the manufacture of steel, his remarks, which are quoted with some slight adaptations, apply to all reduction works: "A skilled manager, amalgamators and engine-drivers you must have, be the output large or small, whether the works run continuously or but a certain part of the time. The difference is that in the case of works with a small output, much of the time of these men, which you must pay for in full whether you use it or not, is either wasted or devoted to work, which in the case of a large output is performed by less intelligent, less skilled, less costly men; double your output and you scarcely need more of these highly skilled men. Most of the additional men are less skilled, many indeed are but assistants of the skilled nucleus which is necessary, and nearly as large in the case of the small as in that of large outputs, and though the foreman or mechanic who is to direct others, must add executive ability to the qualifications he would otherwise need and hence commands higher pay, yet the extra expense thus caused should raise the average cost of the daily wages, much less than it is lowered by the employment of the larger number of relatively unskilled assistants, due to the greater output. There are many trivial and very simple duties which call for little intelligence. With a large output those of each kind recur so often that cheap men can be fully occupied with them. The expensive men in cases of small output, do these trivial, simple acts with their costly labour, which in cases of large output is restricted to the difficult tasks which require it and which fully occupy it. Or, if justified by the greater output on the unit of which their pay forms a relatively small charge, you employ additional skilled and costly men, they are not mere duplicates of those you had before, they bring a different skill and additional knowledge to your aid, permitting economies and devising improvements otherwise unattainable."

This question of skilled labour, it will be noticed, is intimately connected with the production of the mine and capacity of the reduction works.

A large company, with perhaps several hundred men in its employ, requires the highest skilled organization to make it pay interest on

* *Trans. Am. Inst. Min. Eng.*, vol. xix., page 1,120.

a large capital, but the direct opposite may be said of a mine in its infancy, working on a small scale, with a small production, which generally means a small net profit. A staff of highly skilled and consequently highly paid officials would obviously be its ruin. It is therefore, fortunate that it is only when a mine reaches a certain size and depth, with a corresponding output, that the services of a skilled staff become indispensable to commercial profit on a large scale. There is a miners' saying that tributers will make a living where a company will starve, and there is a great deal of truth in it, if only because tributers have no general expenses to bear.

In the mining and milling of gold and silver ores, speaking from the writer's own experience, the skilled Anglo-Saxon workman, whether he be British, American, or Colonial, distances all competitors. Each subdivision of the race numbers in its ranks hundreds of miners (using the expression as covering mill-work as well) second to none in the world, though not a cheap man certainly. The payment of a low rate of wage to the skilled workman is by no means the only, or indeed the most expedient means of lowering the cost of production, since the cost of a day's labour in any responsible occupation is in reality in inverse proportion to the intelligence as well as the sinew and endurance it represents, and in reducing the rate of the one you run great risk of lowering the standard of the other, so long as superior skill can command better wages elsewhere.

The writer's contention is that the cheapest skilled labour is not always, indeed is very rarely, the most economical, if only because the highly-paid man is the most contented, better fed, and better housed man, and the author proposes to show that it is quite possible to cheapen costs without lowering wages, providing a reduction in working expenses is not demanded suddenly, whether by strikes or market fluctuations. To illustrate this,* the Atlantic mine, in Michigan, which produced native copper from 1873 to 1877, scarcely made ends meet, and had to levy calls; since then, with the price of copper steadily declining, and also the yield of the ore, the cost of mining has been so greatly reduced that the company has made a profit every year. The workmen, who earned on an average from £10 8s. 4d. (50 dollars) to £12 5s. 10d. (59 dollars) a month in 1873, continue to earn almost as many dollars, and owing to the greatly reduced cost of living they are now very much better off and save really more money than they did twenty years ago.

* These particulars are taken from a leader on "The Free Coinage Question," in the *Engineering and Mining Journal* (New York), vol. lii., page 497.

The chief items of cost at the Atlantic mine for three years are shown in the following table:—

	1873.	1880.	1890.
Tons treated	51,048	169,825	278,300
Pounds of ingot produced	863,366	2,423,225	3,619,972
Percentage of yield	0·81	0·71	0·65
Average selling price, cents. per lb. ...	26·66	20·55	15·21
Cost of stoping per fathom	$22·28½	$14·35	$4·21
„ driving per foot	19·09½	12·90	5·10
„ sinking shafts per foot	38·03	18·04	22·90
Cost per ton, mining and all surface expenses, taxes, etc.	5·12	2·14	1·04
Cost of transport, 3 miles from mine to mill, per ton	0·15	0·09½	0·03½
Cost of stamping and concentrating ...	1·05	0·38[18]	0·27[78]
Cost of freight, smelting, marketing, and New York office, cents. per lb.	0·34[9]	0·33	0·20[87]
Total expenditure per ton of rock treated	8·26½	2·47	1·67
Net profit per ton of rock treated	...	0·22	0·29¼
Loss do.	4·53	...	...

It would appear from this table, that since wages have practically remained unchanged, or are in fact higher (considering what they will purchase), the reduction in the cost of producing copper is due to improvements in mining, in stoping, in drifting, and through the use of rock-drills, high explosives, and other modern improvements, reductions in freight charges, in milling, and concentrating, in smelting and marketing, due to improved systems of doing the work.

All this shows that improved processes, greater facilities for the distribution of products, and greater skill and economy in the management of business have brought about the lower prices of recent years, and with them have added to the general prosperity.

If you look for the cause of these developments of the practical science of mining, it may be traced largely to technical education, the development of technical societies and technical literature. Mr. John Birkinbine, in his presidential address at the Montreal meeting of the American Institution of Mining Engineers,* states that there are to-day in the United States four engineening societies of a national character, with a membership as follows:—

	Founded.	Membership.
Society of Civil Engineers	1865	1,650
Institution of Mining Engineers ...	1871	2,400
Society of Mechanical Engineers ...	1880	1,650
Institution of Electrical Engineers ...	1886	650

* *Trans.*, vol. xxi., page 962.

A few facts selected from many which could be mentioned, illustrate the progress made during the existence of the American Institution of Mining Engineers from 1871 to 1893. The annual output of iron ore has increased from 3,000,000 to over 16,000,000 gross tons. One and two-thirds million gross tons of pig-iron was the output of the blast furnaces of the United States at the birth of the American Institution. Last year shows a total of over 9,000,000 gross tons, while, owing to improved construction and methods, a smaller number of furnaces yield the larger quantity of pig iron; three-quarters of the American product being produced with coke. The growth of the American steel industry is not less remarkable, and the enquiry might be carried further into the manufacture of rails, plate and bar-iron, and steel nails, cars, machinery, and great works fitted with superb appliances for fabricating them. The statement that a ton of pig-iron, of bar or plate-iron, or a keg of nails now sells at from 33 to 40 per cent. of what was received for it in 1871, whilst the price of steel rails is but 25 per cent. of what these commanded in 1871, is an eloquent commentary on the debt mining owes to the development of scientific principles by mining institutions. In 1871, the greatest depth which had been reached in any of the Lake Superior copper mines was 1,000 feet, and the price of copper stood at 30 cents and upwards, yet it was then impracticable to work those mines which did not produce mineral carrying 2 per cent. or more of copper. At the present time there are mines in the same district 4,000 feet deep, and with copper selling for 12 cents per pound, mineral yielding 0·6 per cent. is raised from a depth of 2,000 feet, crushed, jigged, and delivered at refining works, and sold at a moderate profit on the operation.

The hydraulic elevator and the deflector applied to hydraulic mining are inventions of great practical importance, which only date back, the one to 1870, and the other to 1876, and, under favourable conditions, gold-gravel has actually been hydraulicked for as low a sum as 3 cents per cubic yard. Figures will be found in this paper which illustrate the advances that have been made in reducing the costs of quartz mining, and improving the extraction of gold, etc., by chlorination and concentration.

Pan-amalgamation for silver-ores has been bettered and cheapened, and silver-lixiviation, direct matte-smelting, and the cyanide process are all modern processes born of scientific enquiry, which have made important advances and are likely to bear yet better fruit. Silver-lead smelting has been greatly developed, the so-called practical smelter having given place to the chemist and lead-metallurgist, cleaner and better work is now

done than formerly, lower-grade lead-ores are utilized, and lead-slags made in 1878 are now being re-worked.

Advances have been made in the utilization of electricity, the production of aluminium, and a host of other directions for which the world is indebted to the efforts of members of societies like those named in the United States, the various branches of the Federated Institution of Mining Engineers, which numbers over 2,000 members, the Iron and Steel Institute with 1,500 members, the Institution of Civil Engineers with 6,000 members, the Institution of Mechanical Engineers, of about the same size, and similar institutions in Germany, France, and other parts of the world working for a common object.

Any union for mutual advancement commands esteem, so long as the better element of membership is not hidden or overruled by selfish purposes controlling the administration of affairs to the disadvantage of its mutual progressive features. A past president of the Iron and Steel Institnte, alluding to the visit of that Society to the United States in 1890, expresses, the writer thinks, the mission and achievments of technical associations for the advancement of engineering, and of mining and metallurgical knowledge, in words which may be appropriately quoted:— "The expeditions, through which we meet eye to eye and voice to voice our friendly competitors, to discuss the interests and the scientific aspects of the industry which absorbs us, has been of great personal and national benefit. It is thus we learn how much has been accomplished by persistent and intelligent labour, how much remains to be achieved, and how, by the free exchange of ideas, and of productions, friendly understanding is promoted, and personal acquaintance is built up."

The author's fifth proposition (the influence of quality and price of supplies, etc., used upon the cost of milling) needs no demonstration, but in regard to the sixth and seventh (the power employed and the situation of the works), the Disraeli and New Queen mills will once more serve as illustrations, the former showing an economy both in cost of fuel and transport. (Table II.) This may be partly explained by the Disraeli boilers being of tubular pattern, and being run up to their full capacity, avoiding waste of surplus steam,* and as regards

* At the New Queen and other Charters Towers batteries, Cornish and Lancashire boilers are almost exclusively used, owing to the character of the feed-water which incrusts the tubes; and at the former mill (as is often the case) the engine-power is largely in excess of the immediate requirements, being intended to provide for any enlargement of the plant.

transport, the Rishton works enjoyed the advantage of a tramway between the mine and mill. Of course where water-power is available it offers a large saving over steam. Taking the total actual net difference in the cost of milling at the two mills, viz: 3s. per ton attributable to the natural advantages of location, and other causes that have been pointed out, if the one mill has actually cost let us suppose £3,000 more than the other to erect, it is certain that on a steady output of 40 tons per diem, it would have paid this sum back with interest in less than two years by effecting a saving of £1,480 to £1,780 per annum, or about £125 to £150 per month.

There is a good deal also to be said about the eighth factor in the case (the efficiency of the general management), as it is the master-key to the whole business.

If the workman is to demand and obtain high wages in the future it will only be by employing high-class technical and business skill in the management and conduct of operations, so as to take advantage of every practical scientific improvement which tends to cheapen the aggregate cost of production, whilst duly proportioning wages in all departments to relative efficiency and usefulness, whether in respect of mental attainments or manual dexterity. Unskilled labour can no more hope to compete with skilled workmen living in comfortable circumstances (though they be highly paid), in a field where skill is required, such as mining, than the skilled workman can expect to banish cheap labour from a field where nothing but slight manual effort and intelligence are needed.

We have seen that the difference between the cost of working in one mill as compared with another depends on its design, location, quantity and kind of ore put through it in a given time, the number and character of the external appliances for catching gold or other metal after it leaves the crushing machinery, the cost of fuel, labour, and supplies, the power available, and the organization and *personnel* of the staff, which directly depends on the general management. In regard to this last essential the gross cost of operations is nothing to gauge it by, the only certain proof of efficiency and economy is the ratio which the cost of handling and treating a given tonnage, bears to the percentage of gold or other metal saved when compared with other parallel cases.

Technical skill, combined with business management, in dealing with material and with men is absolutely necessary nowadays for the regulation of work, as the general efficiency of the staff as well as the expenditure on supplies, etc., depends upon it. Discrimination in employing the right man and the proper material in the right place, is the pivot upon

which management turns, though it amounts in the former case simply to a knowledge of the time when to put on an extra man at a profit and when to knock one off the pay-sheet, where to employ a mechanic and where to make shift with a labourer—a knowledge only to be gained by experience of men and materials.

Where you have to deal, as you must in any large mining undertaking, with the three different factors of brain, manual skill, and muscle, it is not less ridiculous for instance to place a man, who may be perhaps an adept at spalling stones, in charge of a mill at the salary of a first-class foreman, than it would be to put the latter to cob ore at the wage of a labourer.

Engineering knowledge and business capacity are co-essentials in selecting as well as in managing modern metallurgical processes and mining enterprises. The capitalist who fails to recognize this "knocks a nail from the inside into his own coffin." The miner who does the same, throws a stone at the only bird which can lay "the high-priced golden egg," where, as in some localities, the sum of his daily wage amounts perhaps to a half-guinea or more.

No one can doubt that the question of mutual and fair adjustment between the two co-important trade factors of capital and labour so as to enable both to exist in harmony, is fast becoming the burning problem of the hour, as it certainly will be of the next decade, or at any rate the next century, and it is certain that the prop upon which the very existence of the miner and the capitalist mutually depends is science practically applied. The miner who thinks otherwise is simply footing his stulls on treacherous ground, which will sooner or later break away with him, and entomb himself and his mates in the *débris*, whilst, as for the guinea-fowl who selects the board-room merely for a roosting place, he will equally promote his own extinction, becoming with the decay of all mining enterprise worth the name like "the dodo."

Messrs. McDermott and Duffield remark*:—"It is not possible to supply the lack of skill and experience by instructions and the owners of mills should take the only safe course in this part of the business by employing a good mill-man who has a record of successful work elsewhere. The owners of a mine or directors of a company may be impressed by the bearing and talk of an applicant for the position, but the frequent and lamentable failures resulting from this method of choosing a mill-man are proof of its inadequacy. Testimonials as to character and experience need just as much examination as the applicant

* *Gold Amalgamation*, page 13.

himself. Past successful record in modern mills is the only testimonial of real value. The salary to be paid a good man should be the last consideration. A cheap man is often a ruinous investment, where the final success of a mining investment rests on the successful treatment of the ore."

Another most important factor of success in running a mine in "loyalty." Loyalty to his immediate chief the miner or mill-hand owes to his shift-boss; the shift-boss to the mine-captain or mill-foreman the same as they owe it to the manager or superintendent, and he in his turn is bound to protect his employers' interests, as represented by his directors. As miners, in whatever walk of life we are placed, whether as employers or employed, we owe loyalty to our industry and to one another; both to those dependent on us and to those we are dependent on. Nothing is more fatal to the proper handling of large bodies of men, whether soldiers, miners, or other operatives, than the lack of *espirit de corps* as well as discipline, and a most unfortunate state of things is often brought about by the exercise of mistaken subordinate judgement in opposition to superior instructions definitely laid down. Rules may be technically right or wrong, but a departure from them is only justified under most special circumstances, which were obviously not provided for when the order was given, but individual foresight, and independent action is of course to be commended, and moreover to be expected in the absence of definite instructions.

There is, perhaps, no business which requires more careful management than custom-milling, for the management is constantly confronted with little mathematical problems like these:—If 20 stamps can be run with 12 men, how many men will be required to keep 25 heads on the drop? It is unfortunate you cannot deal with men in fractions except on paper, but it is sometimes possible to solve the problem with a boy. To take another example:—If it costs you 1s. 3d. to cart 1 ton 3 miles on a fine day on a good road, how much will it cost to bring in stone 4 miles on a bush road on a rainy day? To work this out you can reckon that your carts will sink from 6 inches to 18 inches in black soil or mud, as the case may be, a factor of no small consequence to the result.

Very frequent mistakes in deciding upon proper plant and processes have scattered over the world reduction-works of all kinds of which nothing remains but the ruins, though such failures happily are rarer year by year, as more skill and experience is demanded by mine owners in the selection and operation of plants.

The selection of a process, as well as the conduct of it, should be

entrusted to a trained engineer, instead of (as has been too frequently done) leaving it to an amateur; as well put a miner to pick the best fleece out of a flock of sheep!

In the discussion on Mr. Curtis' paper* one speaker remarked on the impossibility of attaching credit to the judgment of managers, like some he instanced who do not consider it worth while enquiring what they are saving or what they are losing, and totally neglect the question of cost; where the ore is shovelled into the battery with a happy disregard of whether it is pounding itself at times to pieces or not. The author heartily coincides with him, but he hopes this will be remedied by more engineers turning their attention to mining; men who can deal equally with the practical, commercial, and technical scientific difficulties of the business, having been alike through the mining school or college, the mine and the mill, beginning of course as assistants or articled pupils,† as it is a grave mistake to suppose that mining can be learnt without a large amount of practical experience of it, both underground and overground, as well as in the office and the shops.

On the other hand, the author has the highest respect and regard for the self-enlightened, self-educated, practical man, than which latter term none in the mining vocabulary is more widely abused or misunderstood. The author knows, personally, men in Australia and elsewhere whom he has the greatest esteem for, who have never had the advantages of a school training, but who were born, not made, engineers, who, by the force of their natural industry, clear perception, and talents, have deservedly raised themselves to the positions in the profession which they occupy and adorn, and he would take the opinion of such men on special points that they have made the study of a lifetime in preference to any other.

They are men, however, who have their heads screwed on the right way, and are of a totally different calibre from the self-assertive, self-advertized, and self-seeking practical man, to whom he has before alluded, who cannot see beyond his natural horizon and scoffs at progress, because in nine cases out of ten he has been brought up neither as a miner nor as an engineer, but has taken to digging late in life, when all other occupations failed.

The author has said enough to show that what applies to labour and management applies equally to machinery and plant, if efficiency be sacrificed to cost; indeed, he should not wonder if the man who framed the

* *Proc. Inst. Civil Engineers*, vol. cviii., page 135.

† The author refers to the system on which colliery engineers are trained, by articling them to a firm of mining engineers in a colliery district.

proverb "penny wise and pound foolish" was a miner who had in his mind's eye the crumbling skeleton of some modern mining company or other which had paid dearly for such a policy in course of time.

One very general mistake to be avoided in milling, as it has ruined more mining enterprises than any other, perhaps, is the erection of a mill, before there is a mine on the spot, sufficiently proved to warrant such a step, *i.e.*, to keep the proposed works employed. Under such circumstances the possession of a mill is a positive disadvantage, as it means, in most instances, that a certain number of men must remain more or less idle part of the time, unless custom ore can be got to keep the requisite permanent staff occupied; under such circumstances it is generally far cheaper to have the ore treated at public crushing mills, if such exist in the neighbourhood.

The writer does not mean, of course, to dispute that the possession of a mill by a private company which can keep it partly, though perhaps not fully at work, does not tend to cheapen production, but the point is that sufficient stone must be supplied by the mine, or from others in the neighbourhood, to keep a certain minimum number of heads constantly employed, and the larger the number, other things being equal, the greater should be the profit. To illustrate the fact that the possession of a mill, if it can be kept moving, assists in cheapening costs the writer would draw attention to a statement made by Mr. L. W. Marsland.* He says (pages 14 and 15): "The point at which quartz ceases to be payable varies, of course, with the circumstances. Thus, in the colony of Victoria at the present time, quartz yielding 10 dwts. of gold to the ton is found to be highly payable, the lodes being of great size. At Charters Towers, the minimum varies considerably. In the great Day Dawn mines 15 dwts. quartz will pay,† as the ore is comparatively free, and the lode (that is to say, the payable portion of it) is very large. As a rule, however, it is understood that, except in the Day Dawn mines nothing under 1 ounce stone will pay; and in some localities such as the Caledonia, it requires a yield of 2 or 3 ozs. to give payable results, the lode being usually very small, and the rock in the walls, which must be taken away to facilitate working the levels, very hard. With the introduction of improved appliances and methods and the consequent reduction

* *The Charters Towers Gold Mines.* Waterlow Bros. and Layton, 1892.

† According to the Report of the Day Dawn P. C. Company for 1891, the cost of mining and milling (excluding general expenses at Charters Towers, £1,348 1s.) came to £1 14s. 3d., equivalent to about 10 dwts. per ton. Including local and London expenses, the author estimates that the company's total costs did not exceed the equivalent of 11 dwts. $18\frac{8}{21}$ grains on an output of 27,416 tons in 1891.

of the cost of mining and milling, it is hoped that the minimum payable return will some day be reduced to such a point as will encourage the working of the many hundreds of reefs on the Charters Towers gold-field, which at present remain unworked, because of their supposed non-payable character."

This statement reminds one of the early days of California.

Dr. Egleston, page 568, remarks: "Prior to the year 1865, it was considered essential that vein matter should yield at least £4 3s. 4d. per ton to be treated with profit, but the introduction of modern machinery has shown that an ore yielding £1 13s. 4d. to £2 1s. 8d. can be mined at great depths and profitably milled when steam is used; and when high-pressure water can be had £1 0s. 10d. is profitable.* Miners' wages are taken at 12s. 6d. to 16s. 8d. per day (10 hours' shifts). Wood for steam costing from 8s. 4d. to £2 10s. per cord, and water-power from 8d. to 1s. 4d.† per ton of ore treated."

A rough division of the gold-bearing quartz-mines of California into three classes may be rationally made.

First, small veins yielding high grade ore in comparatively small quantities whose cost of extraction and milling is £5 4s. 2d. per ton, and the average yield, say about £10 8s. 4d., as an example we may cite the New River district in Trinity County. The quantity of this ore treated amounts to about 5 per cent. of the ore extracted in the state.

The second class would embrace all the large mines, including 75 per cent. of all the ore extracted in the state. The average cost of mining and milling ore of this class, including general expenses, would be about 12s. 9½d. per ton to mine and mill; the Plymouth and the Zeile, on the mother-lode in Amador County, whose ore only assays £1 5s. 9d. to £1 11s. 7½d. per ton, are instances in point. The ore is free milling, carrying 1½ to 2½ per cent. of sulphides which are often very rich, assaying as high as £20 17s. 8d. to £41 13s. 4d. per ton. Shaft-sinking costs £3 2s. 6d. to £3 15s. per foot. Sawn timber £4 3s. 4d. per 1,000 B.M. Round poles 16 feet long, 14 inches to 16 inches in diameter, 12s. 6d. each. Miners wages are 10s. 5d. per day.

The third class, which comprises 20 per cent. of all the ore mined in the state, would include the Bodie, whose grade is rather high, yielding £6 5s. per ton, but expensive to mine and mill on account of the high

* One might even say 14s.

† This is paid to ditch companies, which bring the water long distances.

prices of labour, fuel, and lumber, the average being about £2 10s. per ton. Most of the gold and silver mines of Ingo County may be included in this group.*

We see from this that the cost of mining as well as milling is largely contingent on quantity, but it is equally to be borne in mind that the quantity milled depends more on nature than on man, who, whilst he can control the amount of development (which is a very important factor in the question) is powerless to alter the size and distribution of the ore-shoots in a vein. Mr. Marsland adds to the remarks that have been quoted "Owing to these circumstances the yearly returns of gold from the field, large as they are, cannot be expected to compare with those of other places in other parts of the world, where, in the great majority of cases, the first consideration is quantity of gold without respect to the question of profit! If the same amount of money were expended in the erection of batteries and developing the mines of Charters Towers as has been within the past few years expended on the South African goldfields there would be no difficulty in producing gold at the rate of 100,000 ounces per month."

Now, although Mr. Marsland's statements no doubt apply to the cost in general of local company-mining on the premier goldfield of Australia, as it is sometimes called, other companies as well as the Day Dawn, can mine and mill below the average Mr. Marsland gives, but mostly the English companies, which possess batteries of their own. The Bonnie Dundee and Mosman for example (both under excellent management) should be able to give a good account of themselves, but in proof of this assertion the author will again take the New Queen and Disraeli for illustrations.

Both these companies possessed mills of their own when the author was in charge of them, and their evidence is the more valuable as regards cost of mining, because they represent two opposite cases in the same locality.

The one (the Disraeli) affording an example of a very wide lode formation, 15 to 40 feet wide, carrying irregular bunches of ore, thrown

* In the Transvaal, some of the large mines on the Randt make 10 dwts. pay. At Barberton, the cost of Sheba gold is said to be £1 4s. per oz. obtained (the ore averaging 1 oz. 12 dwts. 22 grains.) In Mysore (where fuel is costly, and the climate is liable to disorganize the native labour at certain seasons when cholera is prevalent) the cost is said to be about £1 18s. 10d. per oz. of gold extracted (the ore averaging 1½ to over 2 ozs. per ton). The actual mine charges at Nundydroog however were less than this, in 1891 amounting to £2 14s. 8½d. per ton.

sometimes to the hanging, sometimes to the foot wall,* which next to an extremely narrow lode, jambed in between hard walls, is certainly the most expensive and variable class of ground to work. The solid feet of ground that must be stoped to yield a certain tonnage being of course greatly in excess of the ore it would represent were the mineralized portion of the vein regular and compact, which all entails extra cost in winning as well as exploratory and dead-work.

The other case, the New Queen cross-reef, is a fair example of a Charters Towers vein, a well-defined quartz lode, averaging from 6 inches to 3 feet in thickness,† with well-defined walls of syenite.

Of course the actual cost of stoping in any two instances, apart from the size and nature of the lode, length of ore-shoot, quality of labour, general management, and character of explosives used, will depend on the system on which the mine is laid out, the ground is removed, ventilated, drained and supported, and the relative perfection of the haulage and other mechanical arrangements for saving labour.

The cost of development and exploratory work, is also liable to still wider fluctuations, and nothing affects the cost of mining more than the proportion which this bears to the producive work (depending on the nature of the lode, and scale of operations), as it obviously stands in inverse proportion to the tonnage of ore that a given area of ground will yield, and varies with the mining experience and judgment of the management; if the factor of luck in exploitation is excluded from the result. More money can be saved or wasted in fact by good or bad judgment in regard to the probabilities of ground, and the method of proving it, than in any other way; the matter rests solely on the mining knowledge of the superintendent and mine captain, based on a sound or unsound acquaintance with the practical problems of applied geology, as bearing upon ore-deposits of different kinds. A knowledge of those branches of geology which deal chiefly with the igneous crystalline and metamorphic rocks, the distributions and occurrence of ore deposits under different conditions; the phenomena of structure and of faults as they variously affect veins is of the utmost benefit and importance to the metal-miner.‡

The cost of exploration and development§ in the case of the New

* Report of Mr. C. P. Purinton, page 5.

† Manager's Report, August 27th, 1888.

‡ In the exploration of mineral ground, the lack of detailed knowledge such as this is likely to prevent the best advantage being taken of natural conditions in laying out the work, and providing for the most economical extraction of the ore.

§ The distinction between the two terms is determined by the shaft, level, or winze being in ore-bearing (payable) or barren (unpayable) ground.

Queen cannot be given as it has not been published, but we can take it at what ought to be a liberal allowance at Charters Towers under similar circumstances to those above stated.

Assuming that each foot of level and winze driven develops or explores an amount of ground equal to 1 foot in length multiplied by half the depth or length of the block on each side of the drivage, multiplied by the average width of the lode between the next adjacent levels and winzes, or the points where such would ultimately be driven or sunk; at the ruling rates of sinking and driving in the district one may reckon that the cost of development and exploration in a reef of the kind described ought not to exceed 7s. 6d. to 15s. per ton on an output of 270 to 500 tons a month.

The cost of mining 1,086 tons (on an average of 270 tons stoped per month) is given in the author's report, dated January 31st, 1889, published by the New Queen company, as £1 8s. 2d. per ton, to which must be added a charge for general expenses, which may be placed* at, say 2s. 5½d. to 5s. 1¾d., reckoning 270 to 500 tons mined and milled per month.

With regard to the Disraeli mine, taking six months' working, during which 525 tons of ore were won monthly, the cost of stoping may be reckoned approximately at £1 11s. 5d. per ton, but the cost fell to 18s. 6d. per ton, breaking 750 tons a month, during two months in 1887. The underground work was under the immediate charge of Capt. Thos. Blaney, a most competent miner, and all round man.

The cost of dead-work and exploration for the same periods and outputs may be taken at 17s. 2d. to 9s. 3d. per ton, to which we must add, say 6s. 3d. to 3s. 4d. for management and general expenses, which should be a liberal allowance.

The cost in the two cases might be therefore calculated relatively as follows:—

	Disraeli. Ore Mined per Month. Tons. 525 £ s. d.	Tons. 750 £ s. d.
Development and exploitation ...	†0 17 2	†0 9 3
Mining (stoping)	†1 11 5	†0 18 6
Milling 890 tons per month	‡0 9 2	‡0 9 2
General expenses	§0 6 3	§0 3 4
Total cost (estimated)	£3 4 0	£2 0 3
	Dwts. Grs.	Dwts. Grs.
Equivalent in dwts. to the ton ...	18 17$\frac{3}{11}$	11 18$\frac{10}{11}$

*. Company's Annual Statement of Accounts to June 30th, 1890 and 1891.
† See pages ~~218 and 219~~ 220. ‡ See Table II. § Proportions assumed.

	New Queen. Ore Mined per Month. Tons. 270				Tons. 100		
	£	s.	d.		£	s.	d.
Development and exploitation	*0	15	0	...	‡0	7	6
Mining (stoping)	*1	8	2	...	‡1	0	0
Milling 852 tons per month	†0	12	2	...	†0	12	2
General expenses	*0	5	1¼	...	*0	2	5¼
Total cost (estimated)	£3	0	5¼		£2	2	1¼
	Dwts.	Grs.			Dwts.	Grs.	
Equivalent in dwts. to the ton	17	16¼¼			12	7¾¼	

Reckoning a pennyweight of Charters Towers gold, as being worth 3s. 5d. on the average, it will be noticed that in the cases given the cost does not come up to Mr. Marsland's general assumed average, and it would be very considerably less on a larger tonnage mined and milled.

It would be a step in the right direction towards making mining more of a business and less of a speculation, if mining companies more often charged and set aside a fixed sum annually for depreciation of the mine, as well as for redemption of capital spent on plant.

Some of the South African companies pursue this sound and proper business policy on the basis that the ore in sight must redeem the cost of its development, for instance, supposing 135,081 tons standing in sight against £24,814 14s. 3d. spent on development for 12 months, a charge of 3s. 8d. per ton would be debited against the stone milled in the year. This is a good policy since mines may, and often do, in fact, become poorer in depth, which may happen in two ways, either by a diminution in the value of the ore, or a reduction in the size of the ore-bodies.

The mine owner is generally anxious to hurry forward the erection of a mill too precipitately, because he knows it is the requisite first step towards making profit at all, unless he can ship his ore, or treat it elsewhere; but what is the result? In some cases it is found, after a large capital outlay has been thrown away, that no mine, worth speaking of as such, exists, and the glowing reports made by some so-called expert, on the faith of which the money was subscribed, turn out worthless. In others, the money which should have been invested first in mining exploration and development is expended on surface improvements, owing frequently to insufficient preliminary investigation and consideration, and the consequent failure to provide adequate working capital. An insufficient balance consequently being left to open-up what might have turned out a paying property, with the result, that the shareholders, tired of calls without returns, refuse to subscribe more money.

* See pages ~~218 and 219~~ 220. † See Table II. ‡ Proportions assumed.

In many such cases, if the funds raised at first had been expended on the mine, the profit from the sale of the ore would have sufficed to erect the plant afterwards, and the investment would have been saved from failure, or at any rate there would be the satisfaction of knowing that the mine had been thoroughly tested as far as circumstances permitted and found unable to pay, and in such a case a useless waste of capital would be avoided.

It is perhaps a hard matter for an engineer who has to prepare preliminary estimates to avoid running one of two risks. Either he is liable to underestimate, in which case the company finds itself, as already mentioned, without sufficient funds to complete what it has undertaken to do; or he must allow a certain margin at the risk of being considered extravagant. But the latter course is really the safer risk of the two, and it is the best guarantee, in point of fact, that the interests of his employer is his first consideration.

Mr. H. D. Hoskold* remarks, "It will always be found as a rule that to err on the side of excess of size of machinery . . is far better than defect." Alluding to natural or artificial means of draining a mine, he also adds: "The allowance to be made must depend upon the requirements of the case, and the judgment and capabilities of the engineer in charge of the execution of the works, but it is not unfrequently the case that the hands of a good man are completely tied by the control exercised by a board of directors, who, perhaps for the first time, may have engaged in mining. Such interference is most absurd, and occasionally proves very ruinous to the shareholders, because a really good and efficient man could not work under such restrictions." What applies to the matter in question is relevant to many other branches of mining, where capital outlay is involved.

"A few of the failures† were the result of speculations for purposes not strictly honest, but they have all brought mining more or less into discredit as a haphazard investment, when in reality with the same foresight, prudence, and management which is given to other commercial enterprises, mining and milling would pay more than double the legitimate profits of ordinary business." Three things only are necessary, the co-operation of the capitalist, the miner, and the engineer for one common object—to advance legitimate mining. United, their power to do so may be compared with a treble-force detonator, separate them, and their useful effect is reduced to the power of an ordinary cap.

* *Engineer's Valuing Assistant*, page 10. † Dr. Egleston, *Silver*, page 445.

Speaking generally, the proper process to adopt is that which all things considered, will yield the largest possible commercial profit, a point it is the business of the mining engineer to determine; but one thing is perfectly certain, the process not to adopt, is that which leaves any doubt whatsoever, whether there will be any commercial profit at all.

Profit and economy in the conduct of business on a large scale, such as most large mining corporations demand at the present day, requires capital, combined with business tact and organisation. The secret of the success of American mining ventures, lies in six facts:—

(1) Americans treat metal mining as a business, that is to say, they purchase mining property first-hand on an investment basis, not as a mere share-gamble, a condition of things brought about by the efforts of mining men and the mining press, a tendency we have commenced to follow in England only in recent years to any marked extent.

(2) They are not afraid of risking a large capital outlay to obtain a a big return.

(3) They carefully estimate the chances before embarking in a new proposition, and if they lose by their own want of judgment, they do not blame their manager or engineer, unless he is the responsible person, but pocket their experience and try again with hopes and prospects of better luck, aided by better knowledge. The difference in fact between the company promoter and the capitalist, whether English or American, who interests himself in *bona fide* mining, is that one does justice to his property and his employees; the other does not care a red-cent either for the industry or the men engaged in it, so long as he can "boom his stock," a very proper object, no doubt, providing the shares are worth it.

(4) They are open to adopt any improvement demonstrated to be such, no matter what the first cost, so long as it tends to cheapen production in the long run.

(5) They minimise their risks of loss by developing promising prospects, rather than financing worked out mines, and multiply their chances of profit as much as possible, *i.e.*, avoid trusting all their chickens to one hen.

(6) They will bid and give a high figure for the highest technical and manual skill obtainable in any special province where it is needed, and it therefore pays to employ such; a principle which people in other parts of the world with post-dated ideas and less foresight would call extravagance, but which, with good business judgment, is in point of fact, the very essence of economy.

Table IV. summarizes various particulars already given, in connexion with which the author has to acknowledge the kindness and trouble taken by a number of his friends and members of the profession in aiding him with every information in their power, but owing to the difficulty of collecting particulars with regard to actual gross cost of different classes and sizes of plant, cost per ton of treatment by different processes, and losses in treatment, he regrets that the scope of this table cannot be still further enlarged.

The writer will be thankful to receive, and will gratefully acknowledge, any information or criticisms which will enable him to extend, revise, or perfect its individual details.

He ventures to say that at the present time it is impossible for any private individual to get complete particulars of the kind, in twenty or even ten instances that would illustrate each process, if the whole world were prospected for data.

It is to the interest of all mining-undertakings that such information should be forthcoming, because anything that tends to a knowledge of these matters is a step towards cheapening mining generally, to the advantage of each individual company engaged in its pursuit, and the industry as a whole.

It is not too much for shareholders in mining companies to expect an annual statement of the working costs per ton, subdivided under such general heads, as have been mentioned. If such particulars are not obtainable it is usually, let us hope, from unconsciousness of the importance commercially, to the stockholders, or otherwise one is forced to the conclusion that the oversight is due to defective management or business direction, or else that owing to other causes the affairs of a company will not bear the search-light of open investigation.

In the Colonies and America, much is done by the various governments in the way of collecting statistics of working; but without attempting in any way to pry into the private concern of individual profit and loss, there is still room for doing more, as is evident from the paucity of information on some of the points that the author has touched upon.

Statistics can be made complete without being unduly voluminous, and their collection is well worth the attention of English mining companies and engineers engaged in metalliferous mining abroad.

To make them of practical value it is necessary that, however, they should be collected on the same system. The following table is mostly a summary of facts previously stated :—

TABLE IV.

Process.	General Nature of the Ore Treated.	Daily Capacity of the Works.	Approximate Average Cost of Treatment per Ton. Lowest.	Approximate Average Cost of Treatment per Ton. Highest.	Metal or Mineral Saved.	Saving Effected.* Lowest.	Saving Effected.* Highest.
		Tons.	£ s. d.	£ s. d.		Per Cent.	Per Cent.
Concentration	Heavy pyritic ores	70—800	E. 0 0 10 A. 0 1 1¾	0 11 10 0 12 6	Blende & galena. copper... ...	60	99
Smelting	Lead, argentiferous and frequently auriferous? ...	24—96	A. 0 12 6	6 5 0	Silver and gold... Lead	90‖ 85	98 95
Chlorination— Vat	Pyritic concentrates... ...	3½—9	A. 0 14 9½	4 3 4	Gold ... ... Silver	90 60	94 65
Barrel	Pyritic concentrates and occasionally "crude ore" ...	2—4 50—200	A. 0 11 10½	3 0 10½	Gold ... ... Silver	92 60	97 65
Cyanide process	Pyritic gold concentrates, tailings, and crude ore	50—250	A. & S. 0 6 10½	2 10 0	Gold ... ... Silver	65 70	84·5 80
Raw grinding	Pyritic gold concentrates §... Pyritic gold ore	2¾—3 20	S. 1 8 5½ E. 0 8 10	2 16 11 ?	Gold and silver	40 78·7	80 91·7
Washoe pan process ...	Pyritic silver tailings carrying gold	150	A. 0 10 3¾	1 6 0	Gold ... ... Silver	25 75·8	42·5 85·8
Pan-amalgamation: Roasting ...	Complex milling ores of silver	20—240	A. 1 13 4	3 2 6	Gold ... ... Silver	40 65·3	60 94
Pan-amalgamation: Dry... ...	Free-milling silver ores	30—100	A. 1 5 0	2 11 0	Silver	70	85
Pan-amalgamation: Wet... ...	Free-milling silver ores		A. 0 12 6	1 17 6	Gold ... ... Silver	45 60	60 80
Boss process	Combined gold and silver ores	120	A. 0 12 6	2 1 8	Silver and gold...	75	85

A.—America. E.—Europe. F.—Africa. I.—India. M.—Mexico. N.—South America. S.—Australia. W.—Wales. * Careless or ignorant management may of course entail larger losses than those given. † Stamps. ‡ Huntington Mill. § With a small alloyage of silver. ‖ Under most unfavourable circumstances.

TABLE IV.—*Continued.*

Process.	General Nature of the Ore Treated.	Daily Capacity of the Works.	Approximate Average Cost of Treatment per Ton. Lowest.	Approximate Average Cost of Treatment per Ton. Highest.	Metal or Mineral Saved.	Saving Effected.* Lowest.	Saving Effected.* Highest.
		Tons.	£ s. d.	£ s. d.		Per Cent.	Per Cent.
Plate amalgamation—							
(1) With concentration and grinding ...	Ores containing free-gold and pyrites	20—160	S. & I. 0 3 0½	0 14 6	Gold	45	87·6
(2) With concentration and chlorination or smelting of the concentrates	Ores containing free-gold and pyrites	80—130	A. { 0 1 5½ / 0 3 3	?	Gold ... ... / Silver	92 / 70	93·8 / 74
(3) Without concentration	Free-gold ores	70—700	W.A.& F.†1/4½, ‡1/0½	0 15 7½	Gold	70	94
Lixiviation— Russel process... ...	Complex silver ores free from *much* lead or copper	75	A. 1 6 3	2 10 4	Silver, etc. ...	82	91·9
„ raw leaching ...	Tailings (raw or roasted) ...	25—75	A. 0 6 10½	0 19 4	Silver	55	60
Ordinary lixiviation ...	Complex silver ores	50—100	M. 0 15 5 / A. 1 0 6	? / ?	Silver	67	85
Patio process	Rich silver ores, containing traces only of lead and zinc	10—19	M. 5 4 2	5 14 11	Silver ... ... / Gold	70 / 60	90 / ...
Tina process (modified)	Complex silver ores	8	N. 1 8 7½	...	Silver	...	60
Modified Tina & Fondo (Bolivian)	Docile silver ores ... ... / Base silver ores	8 / 8	N. / N. 1 17 6	...	Silver ... ... / Silver	92 / 80	96 / 85
Pyritic smelting ...	Complex silver and gold pyritic ores	100	A. 0 12 6	?	Silver and gold...	?	96

A.—America. E.—Europe. F.—Africa. I.—India. M.—Mexico. N.—South America. S.—Australia. W.—Wales. * Careless or ignorant management may of course entail larger losses than those given. † Stamps. ‡ Huntington Mill. § With a small alloyage of silver. ‖ Under most unfavourable circumstances.

(*To be continued.*)

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THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.*

By A. G. CHARLETON.

Part V.—Gold-milling.

The writer will now examine the differences in details of milling practice which obtain in different parts of the world, considering the local conditions and character of the ore treated, in order to account for the diversities which exist.

General Details and Practices in California.

In this connexion, the writer begs to acknowledge his indebtedness to a paper on the subject by Mr. J. H. Hammond,† in the *Eighth Annual Report of the California State Mining Bureau.*

The California ores are mostly quartzose, carrying free gold and iron pyrites, sometimes accompanied by arsenical and copper pyrites, but more frequently galena and zinc blende. Auriferous tellurides and some of the rarer minerals are occasionally met with, but they are of little economic importance.

The vein-filling is sometimes countrified (consisting of wall-rock more or less altered) but is usually quartz, in some instances associated with calcspar, which, however, but rarely forms exclusively the vein-filling or gangue.

The value of the Californian gold ores varies between 14s. 7d. and £1 13s. 4d. per ton in the low-grade ores, up to between £3 2s. 6d. and £6 5s. per ton in those of high grade; £2 to £2 10s. is probably a rough average of the grade of ore at present treated. The percentage of sulphides (mostly iron pyrites) runs from 1 to 5 per cent. of the ore milled; 2 per cent. being an average approximation in all probability of its pyritous contents.

The saving of these sulphides, though small in quantity, is an important feature in the treatment of these ores, and therefore the majority of the mills of the State have their plants specially adapted to close-saving in this respect, although the value of the concentrates is subordinate to the free gold present in the ore.

* *Trans. Fed. Inst.*, vol. iv., pages 233 and 351; vol. v., page 271; and vol. vi., page 69. † *The Milling of Gold Ores in California.*

The average value of the concentrates being high, accounts for this, and may be put at £16 13s. 4d. to £18 15s. per ton. In the low-grade ores the gold occurs disseminated in particles through the ore, rarely visible to the naked eye. In ores of high-grade it is often found massive and sometimes in laminæ along the planes of division of the quartz (ribbon-rock), while in other cases it assumes the form of wire (filiform), and is also occasionally arborescent.

Specimen ore is often sold to jewellers, who pay from £4 3s. 4d. to £5 12s. 6d. per ounce for the free gold that the quartz contains. The pyrites is generally massive, though sometimes found crystallized, but iron pyrites of the latter character rarely carries much gold. The sulphides in the country-rock are likewise of little value.

In some of the Californian mills the rock-breakers are arranged in a way somewhat out of the common, but which is a very excellent one. The ore coming from the mine is discharged in the usual way on to a grizzly which separates it into two classes; what passes through goes to the main ore-bin (underneath at the back of the battery), while the coarser rock is retained in what is called the coarse ore-bin, behind the rock-breaker. The grizzly is set so as to form the bottom of the upper half of this last-named bin, the lower part and sides being boarded in.

A shoot leads from the door of the coarse bin (which is worked by a rack-and-pinion) into the jaws of the rock-breaker. By this arrangement the delivery of the ore is controlled, ensuring an almost continuous supply of stone to the machine, which greatly increases its capacity and reduces expense, as it saves the labour of a feeder which is necessary, where, as in most mills, the ore is discharged over the grizzly on to the rock-breaker floor, and also maintains a regular feed.* The only point to be observed is that the stone is sent to the mill spalled to a size which will all enter the jaws of a given-sized crusher.

At the North Star mill with this arrangement, one (15 inches by 9 inches) breaker crushes 30 to 40 tons of hard rock in 5 to 7 hours, effecting a saving in wages of two or three men as compared with the labour required ordinarily, while evidence of its uninterrupted work is shown by the fact that it requires 12 instead of 8 horse-power as is usually computed. The crushed ore joining the fines which have passed the grizzly, is well mixed, which ensures uniformity in the charges fed to the stamps.

* In some of the newer German concentration-works a reciprocating-table, at the end of the shoot, actuated by an eccentric, is employed to ensure this result.

Where fall permits, it may sometimes be found advantageous to crush coarse in one rock-breaker, and deliver the product to a second one, breaking it finer. This greatly increases the capacity of the stamps. The rock-breakers are usually set to crush to 2 or 3 inches. The shoes and dies last from six to eight months; and when made of steel they run about twice as long.

Self-feeders.—The use of self-feeder machines saves a large amount of labour, increases the capacity of the battery from 15 to 20 per cent., besides effecting a considerable reduction in the wear of screens, shoes, and dies.

Other conditions being equal, low feeding increases the capacity of the stamps, that is to say, the ore should be fed in regularly, and in small quantities at a time. When thrown into the mortar irregularly in large charges, the ore is piled up in the box, and the stamp cushions upon it, which impairs the effectiveness of the blow. The best hand-feeding is superior to mechanical devices, but men are mortal, and either through ignorance or inattention, caused by the extremely trying conditions under which such work must be performed for long spells at a time, the machine on the whole is the most reliable, apart from the monetary question of saving in labour where labour is dear.

Automatic feeders are of various types, those with a revolving-carrier (such as the Hendy Challenge) deserving the preference for wet, clayey and sticky ores. Those with a shaking-table, such as the Tulloch and the Stanford roller-feeder, give satisfaction, however, with certain classes of ore, and being the cheaper, are frequently employed. The Challenge feeder costs £52 1s. 8d. in San Francisco, occupies 23 cubic feet packed for shipment, and weighs 750 lbs.

The Hendy improved Challenge suspended-feeder does away with the underneath framing of the ordinary type of machine, and renders the feed-side of the mortars more accessible, the machine being supported on parallel tracks overhead, between the battery-posts and ore-bins. It is specially adapted for this reason for feeding rolls.

Mortars.—This important feature in battery construction depends on the character of the ore and deserves more consideration than it generally receives. Narrow mortars accelerate the discharge of the pulp, and are therefore disadvantageous, where it is desired to amalgamate on inside-plates, or settle coarse-gold in the box. They also occasion an excessive breakage of screens, adding to the expense of renewing and changing them, if the ore is of a hard flinty nature.

The liability of breakage can be reduced by raising the height of the discharge, but this counteracts the advantage aimed at in using a narrow mortar, viz., large capacity. Each mortar is fitted inside, with side and end-linings of cast or wrought-iron to protect it from wear. These linings when of cast-iron last six to nine months. In some mills they are held in place by bolts with the heads countersunk in a recess in the plate, which are tightened up from the outside; a better method, however, is to halve the plates together at the ends.

In many Australian batteries, a length of about 9 inches is cut out of the middle of the front-side of the mortar, and replaced by a dove-tailed iron casting of the same shape and thickness as the section removed. This slides up and down (bearing against a shoulder) when the screen frame is taken out, and facilitates cleaning the box at a general clean-up. When the sides and bottom of the false-section are properly planed and the opening trued up to correspond, the mortar remains perfectly watertight.

The height of discharge, *i.e.*, the vertical height of the lower edge of the screen above the die, should be regulated to correspond more or less with the width of the mortar, taking into consideration the object aimed at in the treatment of any special ore. Narrow mortars require a relatively higher discharge to avoid breakage of screens, and to prevent scouring the inside copper-plates. Having fixed on the proper height of discharge, it is most important to keep it uniform. There are various ways of doing this, which are referred to elsewhere. With ores quickly crushed and readily discharged, it is advisable to raise the height of the discharge, to retain the pulp in the mortars long enough for its proper amalgamation. Where an ore contains a high percentage of sulphides, and cannot be advantageously amalgamated in the battery, or where stamps are used with a view to large capacity in crushing, and it is desirable at the same time to catch the coarse-gold liberated, by allowing it to settle in the box, or in the third case of an ore containing brittle sulphides, which from being subjected to lengthy stamping are liable to be slimed, double-discharge mortars may be found advantageous. Stamps crush faster than the screens ordinarily discharge the pulp, with the result that many of the particles are subjected to unnecessary pulverization, sliming the sulphides which can with difficulty be settled, producing float-gold, and when coarse occasionally perhaps pounding it; hence double-mortars sometimes serve a necessary purpose. Their objections and limitations are: (1) where copper-plates are used inside, mortars of this kind are inconvenient, and are too roomy for efficient amalgamation; (2) owing to the extra water

they use, they are prejudicial to the outside plate-amalgamation, and in some cases to the subsequent concentration, depending on the character of the concentration-method, and the machines adopted to that end. The feed-opening of a mortar should extend nearly the whole length of the box, and be about 4 inches wide, allowing for the upper part of the back-lining.

Screens.—In California, various kinds of screens are used—steel and brass wire-cloth; slot and needle-punched (tough but soft), planished Russian sheet-iron (which has a smooth glossy surface), and tin. These last are not so thick as Russian iron and consequently admit of more rapid discharge than Russian iron for the same diameter of perforations, but they do not last so long. The numbers by which steel and brass wire screens are designated correspond with the number of meshes per lineal inch. The most common sizes in use in California are 30 and 40, No. 30 being made of No. 31 wire and 40 of No. 33 wire, the latter costing in San Francisco 1s. 6d. per square foot. Each battery takes 3 to 4 square feet.

The screens wear most at their lower edges. Brass wire-screens last 10 to 14 days. High discharge and wide mortars promote the life of the screens. One brass wire-screen, No. 30, bar accidents, will put 120 to 140 tons of ore through, before it is worn out. Steel wire is liable to rust, and is therefore not much in favour.

The area of discharge in wire-screens is much greater than in slot or punched iron, producing a more uniform pulp, hence it is in a condition particularly adapted to after-concentration. The numbers and sizes of needle-punched and slot screens correspond to the number of the corresponding needles which will pass through the openings. Nos. 5, 6, 7, 8, and 9 are the commonest sizes. The slots are horizontal or diagonal (angle slots). They are generally burred on the inside to prevent clogging, as the opening enlarges outwards, but this tends obviously to reduce the discharging capacity of the battery, from the fact that a particle of ore which does not strike the opening straight, has no chance of getting through; that is to say, either obliquely, or when flowing down the grating.

Russian iron screens last 15 to 40 days, a month being about the average. The screen-frames are of wood, and should have a batter of about 10 degs. projecting outwards at the top; they rest in a recess cast in the front of the box, and are held in place by wooden wedges driven under claws cast on the mortar. A strip of blanketing should be laid round the frame.

A piece of heavy canvas is generally hung across the front of the mortar.* Stamping should be carried just far enough for the proper liberation of the gold and sulphides from the veinstone. When the gold is finely divided, as generally happens in low-grade ores, the stamping must be proportionally finer than when the gold is coarse. When the sulphides constitute an important factor in the value of the ore it is desirable to crush coarse to avoid excessive loss by sliming them. That dead-stamping is a serious and by no means visionary fact is attested by the statement that generally over 80 per cent. of the ore discharged through a No. 30 screen will pass a No. 60, and often 50 per cent. will pass a No. 120.

TABLE OF SCREEN SIZES.

No. of Needle.	Corresponding Mesh.	Width of Slot. Inches.	Gauge of Russian Iron. No.	Thickness of Iron. American Gauge. No.	Weight per Square Foot. Lbs.
5	20	0·029	14	23¼	1·15
6	25	0·027	13	24	1·08
7	30	0·024	12	24½	0·978
8	35	0·022	11	25	0·918
9	40	0·020	10	26	0·827
10	50	0·018	9	27	0·735
11	55	0·016	8	28	0·666
12	60	0·015	8	28	0·666

Russian iron is sold in sheets measuring 28 inches by 56 inches, equal to 10·88 square feet. Some manufacturers' measurements differ somewhat from the table. The Attwood screen-measure is very convenient for determining the size of orifices. For inside battery amalgamation, the mortar is provided with strips of silver-plated copper-plate, back and front, seated on wooden blocks (provided with iron-facings where the keys come outside), resting on top of and flush with the inside liners (an Australian term for the lining-plates). Silver-plated brass screws are used to fix them on.

Drop of the Stamps.—This depends on the character of the ore, the speed and weight of the stamps, and the duty required, varying from 4 to 9 inches; 6 inches being about the mean in California. Sufficient drop must be given to produce a good splash. Soft and highly mineralized ores need a low drop usually. The order of drop is of importance, as on it depends an even distribution of the pulp on the several dies. Adjacent stamps should never drop consecutively, as this occasions accumulations of pulp in the end of the mortar, by which the efficiency of the stamps at

* In Australian batteries, a cover of sheet-iron or wood, hung on hinges (cast on the top front of the box) is used.

that end is reduced, whilst those at the other extremity are liable to pound iron. The order, 1, 4, 2, 5, and 3 gives a good splash, and gives satisfactory results. 1, 5, 2, 4, and 3 is also extensively adopted. The remedies, for a bad order of drop, are either to change it, re-cutting the key seats in the cam shaft (by no means a simple affair), or else to suffer the misfortune of having to give one stamp a greater drop than the others.

Duty of the Stamps.—This term, which is applied to the quantity of stone crushed per head by the battery in 24 hours, will depend on the weight of the stamp, number of drops, height of drop, height of discharge, size of screen-mesh, area of screen-opening, dimensions of the mortar and character of the ore. Clayey and hard ores reduce the duty. 2½ tons per stamp in 24 hours is about the average duty of Californian batteries.

Speed of the Stamps.—Low speed is necessary with heavy stamps and high drops. In California, 900 to 950 lbs. stamps, with 6 to 7 inches drops, are run at 85 to 95 drops per minute. With double cams, the speed must not exceed a certain rate, for if the revolution of the cam does not give sufficient time for the stamp to fall, the descending tappet striking the toe of the returning cam, is very likely to cause a breakage or dislodge either boss, shoe, or tappet. A fast drop produces a good splash in the mortar.

The cam-shaft is generally made of mild steel, and the cams themselves may be made of hard chrome steel. The curve of the face should be a slightly modified involute.* The cam-shaft pulleys are best built up of wood, thoroughly seasoned, and the joints filled with white lead and oil. They are held by 3 feet cast-iron double sleeve-flanges. The hub is bored and fitted to the cam-shaft and fastened with a steel key. The outside flange is bored and fitted to the sleeve with a jib-headed steel key. Kiln-dried sugarpine is a favourite material for making these pulleys. Cams ought to last several years. Tappets should have both faces turned true, and be fixed on with a steel jib and two steel keys, bored accurately with the jibs in place, to fit the stems, and counterbored opposite the jibs by moving the centre ¼ inch, taking a cut ⅛ inch deep, with a diameter ⅛ inch less than the bore. This gives three bearing-points and allows the tappet to move easily on the stem.

* Split cams in case of breakage are more handy to change than solid ones.

The working-face, with the exception of a small annular ring next the stamp-stem, is subjected to equal wear, hence this central part is recessed out $\frac{5}{8}$ inch to $\frac{1}{2}$ inch wide and deep on both ends, so that the cam may not be worn down by the projecting ridge which would be liable to be formed on the inside of the tappet. Where one face is worn, the tappet is reversed. The tappets, like the cams, are usually made of hard steel.

The revolving cam in lifting the tappet causes it to partially revolve, communicating a certain amount of rotation to the stamp as it drops. Its chief importance is in equalizing the wear of the shoes and dies, which promotes better crushing. With an ordinary amount of grease on the cam, a stamp should make a complete revolution in four to eight lifts, but if there be an excess of grease, or the tappet has been allowed to become grooved, the rotatory motion is much impaired. The average life of the tappets is four to five years. They may be broken by being keyed too tight. When their faces are worn they are planed down. The importance of keeping the height of discharge regular is second only to maintaining a fixed height of drop. A frequent examination should be made to see that the cams have not shifted, which is very liable to happen if the tappets are not properly secured. The cams must be greased sufficiently, but not in excess; nothing is more disgusting or liable to interfere with amalgamation than grease being thrown on to the aprons or falling on the boxes.

The wooden covers of the mortar are no protection against oil, as the writer has seen it in some mills running down the stems; he may say that they were not American mills. Wooden finger-bars, fitted into latch-sockets lined with leather, hung on a wrought-iron jack-shaft at the back of the battery, are mostly used in America to hang up the heads. To perform this operation neatly, requires some skill, the battery-man standing on the platform at the back of the stamps just below the jack-shaft, and inserting a wooden lever with his right hand under the tappet as the cam comes round. This lifts the tappet higher than usual, allowing him to slip the point of the finger-bar underneath the back of the tappet, before the cam drops the stamp.

In Australia, a different method is generally employed. Two flat straps of iron are fastened to the battery-posts horizontally, a little higher than the full height of the throw of the cam (one at the back and one in front of the stamps). Resting across these straps are five rectangular bars of iron attached to each battery, provided with a hook at one end and a handle at the other. The hook slides on the front strap, and

the battery-man, taking the handle at the back in his hand and resting a short crowbar on the back strap, catches the tappet as it is raised, prizes it up slightly, and slips the cross-bar sideways underneath.

Shoes and Dies.—These are either of iron or steel. Ordinary cast-steel used for this purpose has a great tendency to chip and cup; recently, however, chrome-steel has come into the market, and in remote districts, where transport is an important item, it has replaced iron.

Steel shoes with iron dies wear more evenly than if both are steel. The life of steel is about two-and-a-half to three times that of iron shoes and dies, while the cost is about twice as great. The mixture of old chrome-steel shoes and dies with iron, produces shoes and dies which wear considerably longer than iron, and where there is no other means of utilizing the old castings this plan may be advantageously introduced. Old ordinary iron castings are sold to local foundries for 1½ to 2 cents per lb.

The shoes are made to fit into the stamp-head by being tapered so as to fit into the core of the socket, the shank of the shoe being surrounded with a number of dry hard-pine wedges attached to it with a piece of twine, upon which the boss is driven on by allowing the head to drop on the bare die. In Australia, in some mills, in place of wood-strips the shank is covered with canvas or blanket in strips, crossed at right angles, overlaying one another alternately, and tied round with thread. The heads or bosses are accurately bored at the top to receive the tapered end of the stem, and have a conical socket cored out at the lower end for the shank of the shoe. Transverse rectangular key-ways, 1 inch by 2½ inches, cross the centre of the head at right-angles, passing through the bottom of the recesses, in which the stem and shoe-shank fit, for the purpose of wedging these latter out with loosening-keys.

The durability of the shoes and dies is affected by the weight, speed, and height of drop of the stamps, the material they are made of, coarseness of stamping, and height of discharge, as well as the manner of feeding the ore and the hardness of the stone. Iron shoes of good quality will last 30 to 47 days; they are worn down usually to from 1½ inches to 1 inch in thickness, and then weigh about 20 to 50 lbs. The dies being protected by the bed of ore that covers them to a depth of 1½ to 3 inches wear longer, but when the length of the shoe is greater than the height of the die the actual life of the latter may be the shorter of the two.

The consumption of iron for shoes and dies per ton of ore crushed is from 1½ to 3 lbs. To obtain a maximum duty, the dies should be kept as high as is compatible with the safety of the screens, and with successful

battery amalgamation; if such be practised. Uniformity in the level of the dies is important, for should one die project much above the others no pulp will remain on it, and the shoe will consequently pound on the naked die. Stems are made tapered at both ends so as to be reversible, and are usually of mild steel. When broken they can be swedged or planed down, and additional lengths welded on if necessary.

Foundations, etc.—Nothing about a battery is more important than well-constructed solid foundations. The mud-sills and battery-blocks are generally made of yellow-pine, free from sap, or sugar-pine. Sugar-pine or red-spruce is used for the framing of the battery. Sun-cracks are filled with melted sulphur or Stockholm tar, laid on while hot. They should be bedded on concrete or on the clean bed-rock. An overhead travelling-crane on a suspended track should extend the full length of the battery. The battery-blocks for each mortar in California are generally made in two pieces each 30 inches thick, and wide enough to fill the space between the line-sills and battery-posts. They are secured with oak keys 4 inches wide, tapered from 6 inches in thickness at the head to 5 inches at the point as well as with six 1 inch bolts. Yokes of 10 inches by 10 inches timber are fitted and bolted to the line-sills and battery-posts. In loose ground the sides of the pit for the battery-blocks may be walled in at the sides and ends. Once the whole set of blocks is level sighted along the whole line (which is generally effected by nailing a board along the front, a short distance below the top, and sawing the projecting part off), the top is planed smooth, making it about $\frac{1}{16}$ inch hollow to prevent it from becoming rounded. It should be covered with sheet rubber $\frac{1}{4}$ inch thick, or tarred mill-blankets and sheet lead ($\frac{1}{8}$ inch thick) to form a seat for the boxes. Personally, the writer prefers the former. The mortars must be levelled true, both lengthways and across, inside.

Amalgam.—In battery amalgamation, the largest proportion of the amalgam is caught on the inside-plates, representing, however, a greater percentage if the gold be coarse, than when it is finely-divided. Width of mortar, height of discharge, nature of plates and attention given to them, and fluidity of the amalgam, all influence the ratio between what is saved inside and on the outside-plates. Sometimes from these causes more is caught without than inside the battery; generally speaking, however, in California 50 to 80 per cent. comes from the battery. The fluidity of the amalgam is generally caused by overfeeding quicksilver, but in tropical climates it is to some extent influenced by the season; being more fluid in hot weather.

The battery-amalgam is invariably richer than the plate-amalgam, its value increasing with the coarseness of the gold, and as might naturally be supposed decreasing with finely-divided alloyed gold. At the Original Empire and Star mills in Grass Valley, California, while the battery-amalgam averages £1 15s. 5d. per ounce, the plate-amalgam only runs 18s. 9d. Amalgam varies greatly in different clean-ups with ores from the same mine. The frequency of cleaning-up depends on the richness of the ore and local practice. The richer the ore, the more often it is necessary to clean-up.

The outside-plates and sluice-plates are usually cleaned up every 24 hours. The amalgam and skimmings are ground with the addition of quicksilver in the clean-up pan to soften and clean the amalgam. The surplus mercury is expressed and the balls retorted with those from the general clean-up. Ten to fifteen minutes are required to clean up each battery, the stamps being hung up and water turned off. The general clean-up is a fortnightly or monthly affair. Two batteries are hung up, the outside battery-plates and screens removed, and inside-plates and dies taken out. The inside-plates are laid over the sluice-plates and the amalgam scraped off, taking care not to scratch the copper. Where the outside-plates are not removable, they must be protected, after they have been cleaned, by boards laid across them before attempting to clean up the mortar. The linings of the mortar and dies must be carefully washed in a tub, and scrubbed down with a brush before replacing them. If any of the castings are imperfect they should be carefully probed. The battery-bottoms are carefully shovelled into pails and removed for treatment, and after sifting out the coarser part of the uncrushed ore, the balance, consisting of several pans full of medium, coarse, and fine ore, mercury, sulphides, amalgam, pieces of iron, steel, etc., from the different batteries, are run through the last box cleaned-up. The bottoms remaining in it, when these have been finally worked down are taken out, and the bulk of the worthless material panned-off, extracting the steel and iron with a magnet; or the bottoms may be run through a tom. What remains is then put through the clean-up pan. In large mills, the barrels and clean-up batea are used. The clean-up pan should be in a special room as close as possible to the battery, the floor being of cement to prevent loss of amalgam and mercury. It should contain two cast-iron tanks, 4 feet long, 3 feet wide, and 3 feet deep to prevent loss and leakage in panning-off. It is a good plan also to give the floor a slight slope towards a small tank cemented into it, to catch any mercury accidentally spilled.

In one corner is placed the clean-up pan before mentioned, (usually about 2½ feet in diameter), provided with hard iron drags.

The final residues from the battery are ground in this pan with mercury, the refuse run off, and the amalgam collected after one or two hours' grinding.

Three men can clean up a 40 stamp mill in this way in 5 to 7 hours, at which time advantage is taken of the stoppage of the stamps to make needed repairs, and to change shoes, dies, screens, etc., if required.

It takes from two to four hours to retort the amalgam in the silver retorts used in large mills. Small mills use the cup-shaped retort. The resulting bullion is weighed and the gold melted in plumbago pots, taking 1 to 2 hours. Borings are then taken, or the bar is chipped in several places, and several bullion-assays are made to determine its fineness.

Grade of the Plates.—The inclination given to the inside copper plates is very variable; for the outside plates, it varies with the sulphides in the ore, the amount of water used, and the fineness or coarseness of the gold. Sufficient grade is necessary to prevent the pulp settling on them. High sulphides and coarsely crushed ore requires a maximum grade. The frame supporting the plates, should therefore be constructed so as to admit of adjustment to suit the ore treated.

The usual grade of outside plates is $1\frac{1}{2}$ to 2 inches per foot. The apron-plate is generally given a grade of from $\frac{1}{2}$ to $1\frac{3}{4}$ inches per foot, $1\frac{1}{2}$ inches being about the average. The sluice-plates have usually a fall of $1\frac{1}{4}$ to $1\frac{1}{2}$ inches per foot. The end of the sluice-plates should be furnished with a mercury-trap. With most ores, steep grades and a minimum of water are to be preferred to lessening the grade, and using more water.

By this plan the gold is rolled, rather than swept along by the water, and better contact is secured.

Shaking-plates suspended on movable springs are an important adjunct to the system of amalgamation; they should be the same width as the sluice-plates, but set at a slightly less grade; the movement of the ore, being assisted by the longitudinal shaking movement given by the eccentrics which actuate them.

There should be one or two drops from 2 to 3 inches in height in the line of plates; a crater-like deposit of amalgam accumulating where the pulp drops from one step to another.

The frames of the plates, with the exception of the apon-plate, which is sometimes supported on a casting bolted to the mortar,* should rest on

* If the more common method of supporting the apron-plate like the lower ones is adopted, the end board ought to be brought well under the lip of the mortar, but kept from direct contact with the battery-blocks and the under side of the box.

bearings independent of the battery framework, to avoid the jar which would otherwise ensue. The plates are held down by wooden cleats at the side.

Mercury is charged every hour or so into the mortars, the quantity depending on the grade of the ore and physical character of the gold. Finely-divided gold requires more mercury than when it is coarse. Sulphide ores also require larger charges, as ores of that class have a scouring effect on the plates and carry off the quicksilver.

The condition of the outside-plates is a guide as to whether too much or too little is being used. The amalgam should be sufficiently pasty to adhere to them, but neither too dry nor fluid enough to roll off.

From 1 to 2 ounces of quicksilver ought to be added to the battery for each ounce of gold contained in the ore. The value of quicksilver fluctuates considerably.

The loss of quicksilver per ton of ore is very variable where amalgamation takes place in the battery. Sulphide ore, especially such as carries galena or arsenical pyrites, occasions a large loss of quicksilver. The large losses in the early attempts to amalgamate the mispickel ores of Marmora, Canada, is a striking case in point.

Loss of gold accompanies the quicksilver, but its amount is liable to be overestimated by some mill-men. The loss of mercury at the Empire and North Star mills is said to exceed that of any mills in California, but *per contra* there are few mills that save such a high percentage of the gold in the ore. Their loss is reckoned to be often as great as 1 ounce of mercury per ton of stone crushed. In other instances it varies from $\frac{1}{8}$ to $\frac{3}{4}$ of an ounce per ton; $\frac{1}{2}$ an ounce per ton or 1 lb. per 32 tons of stone crushed is about a mean. The general causes of this loss are referred to elsewhere.

Concentration.—The pulp in most Californian mills is conveyed from a distributing-box at the end of the copper-plates through 2 inches pipes to the concentrators. These have various devices for saving amalgam, free-gold, and quicksilver, that may have escaped the preceding appliances, as the pulp delivered to the concentrators ought not to carry these substances. Any quicksilver, lost by previous careless handling or imperfect amalgamation, that finds its way to the belt-concentrators is lost by volatilization in roasting the concentrates preparatory to treatment by chlorination, which is generally practised in California; and if the gold be coarse, whether it be free or amalgamated, it introduces difficulties into the chlorination treatment, hence the necessity of saving these substances before they get to the concentrator-belts.

There is no preliminary sizing of the particles of ore in Californian gold-mills, and therefore the conditions essential for perfect concentration are wanting. Despite this serious disadvantage some of the concentrators, such as the Frue-vanner* and Embrey, succeed in obtaining clean concentrates with but little loss of auriferous sulphides. Other concentrators in use are the Golden Gate, Duncan, and Hendy pan. The Frue-vanner and Triumph are, however, the most popular. Each has features of superiority, and each its advocates. Both have quicksilver and amalgam-saving devices.

The construction of these machines is too well-known to require to be detailed here. The author will only allude to one or two special points that may be noticed in their arrangement.

At the Empire mill a special arrangement is used for removing the sulphides from the belt and depositing them conveniently for shovelling into wheelbarrows. The specifically lighter particles are carried down the surface of the belt by the current, and pass as tailings to blanket-sluices outside the mill. Fine particles of heavy specific gravity have a tendency to cling to the belts, which promotes their recovery.

There should be a separate tank outside the mill into which the overflow from the tanks below the belts is conducted, so that the fine sulphides in suspension in the water may not escape. The consistency of the pulp and its even distribution across the belt must be carefully regulated. The speed at which the belt travels is also very important, varying from 3 to 12 feet per minute in the Frue, and 3 to 4 feet in the Embrey. The grade given to the belts should be properly adjusted, and the surface must be level cross-ways.

The clear water jets which dilute the pulp require ½ to 1 gallon of water per minute. The depth of pulp or load on the belts is from $\frac{5}{16}$ to ¼ inch, and the feed-water amounts to 1 or 2 gallons. The capacity of the machines varies with the character of the ore; ores carrying a high percentage of sulphides, or mineral in a fine state of division (slimes) require more concentrators than when they are of an opposite character.

Two concentrators are the usual allowance for a battery of 5 stamps. A floor-space of 20 by 10 feet should be provided for each concentrator, and if a double row be used, they should be set head to head in front of the batteries, with a passage-way of 5 to 6 feet between them.

A point of the utmost importance is to secure intelligent supervision and regularity in running, and, to do this, power to run them should be

* The writer believes he may claim to have run the first automatic feeders and Frue-vanners which were operated successfully in a North Queensland battery.

supplied by an independent motor; for when the concentrators are connected with the main driving shaft, the stoppage of other machinery (such as a rock-breaker), has a disastrous effect on the work they do.

The number of oscillations given to the machines is likewise of consequence. With the Frue, it varies from 180 to 200 revolutions per minute, with a throw of 1 inch. In the Embrey, it is 230 revolutions. Three men, one head concentrator, and two assistants can easily attend to sixteen concentrators, *i.e.*, for a 40 stamp mill per 24 hours. The duty of the former is to supervise, repair, and oil the machines while the assistants rake out and remove the sulphides to the sulphide room.

In a large mill with 80 stamps, it is preferable to employ one man per shift to attend solely to the adjustment of the machines, and an engineer to make repairs for all the machinery about the mill. One roustabout can remove the accumulated sulphides. Adjoining the concentrating-floor, on a level with it and on the sunny side of the mill when practicable, there should be a room to store the sulphides, with a concrete floor to drain off the water to a central tank towards which the floor has a slight slope. It should be well lighted so as to dry the concentrates in the sun.

The concentrates in California are generally treated by the Plattner chlorination process (unless the mill is large enough to have chlorination works of its own) at custom's establishments. These establishments charge about £4 3s. 4d. per ton for treatment, and guarantee a return of 90 to 92 per cent. of the assay value of the ore.

Blanket-sluices.—The length of the blanket-sluices outside the mill is governed by the value of the tailings. From 100 to 200 feet usually suffice. These sluices have a grade of 1 inch to 1¼ inches per foot. The sands collected on the blankets are generally ground in a pan, with a diameter of 3½ to 4 feet, like a silver-mill pan. The pulp leaving the sluices constitute the mill-tailings, and to keep a proper supervision over the work, samples should be taken of those several times during the day and night, and their value ascertained periodically.

Mill-men are, on the whole, exceedingly remiss in this respect. An automatic sampler ensures the work being done properly. The McDermott and Starr machines are used for this purpose. The pulp is assayed three times a week, the samples being examined carefully to see to what cause the loss is to be ascribed. Microscopical examination of the sands should be made occasionally to ascertain if a perfect liberation of the free-gold from the gangue has taken place. Thorough and regular sampling is a check on the mill work, and causes the men to be alert and zealous in discharging their duties.

The wooden batea and miner's gold pan are used for panning-off.

The average value of the ore milled is represented by the value of the gold saved plus the value of the sulphides caught, and the value of the tailings. The percentage of gold saved by the mill is calculated from these factors, and represents the efficiency of the process. Other things being equal, it will vary with the class of ore, being less with brittle sulphides and fine-gold than when the opposite more favourable conditions obtain.

Most of the loss occurs through loss of sulphides, consequently a large percentage of rich sulphides, is liable to produce rich tailings. There are few ores in California, however, from which 80 per cent. and upwards of the assay value, cannot be extracted by careful mill-men in well arranged mills. The majority save 75 to 85 per cent. Exhaustive investigations, extending over 18 months, at the North Star and Empire mills, show a saving of 82 to 94 per cent. The usual percentage, according to reliable semi-monthly returns, is 86 to 90 per cent. In these estimates no deduction, of course, is made for any after loss in treating the sulphides; but this is usually unimportant, as before stated.

The mill labour in a 40 stamp mill per 24 hours is as follows:—

	£	s.	d.
One man at rock-breaker, at 10s. 5d.	0	10	5
Two amalgamators, at 12s. 6d.	1	5	0
Three concentrators, one at 12s. 6d. and two at 10s. 5d...	1	13	4
	£3	8	9

The rock-breakerman also attends to the blanket sluices, and is employed in other work about the mill. Where steam power is used, two engineers and one man to pile wood near the boilers, would be required in addition to the above staff.

The cost of milling per ton in a 40 stamp mill, with a capacity of 80 tons per 24 hours, driven by water power, therefore stands as follows:—

	s.	d.		s.	d.
Mill labour as above	0	10½	to	0	10½
Assaying, retorting, and superintendence	0	1½	„	0	1½
Supplies	0	3½	„	0	5
Quicksilver	0	0½	„	0	2
Lubricants, screens, illuminants, machinists' time, etc.	0	2	„	0	4
	1	5½	to	1	10½

To this must be added the variable cost of water-power. The use of steam-power would add about 5d. per ton to the above estimate for labour, with ½d. additional for repairs, lubricants, etc., incidental to using steam.

An electric plant to illuminate mill and offices costs about £125, the cost of producing the light being but little beyond the cost of extra power to run the dynamo. Good illumination is very desirable in a mill.

The charge for assaying, retorting, and superintendence is based on the salary of £25 per month for a man to perform these duties in addition to rendering other services, as clerk, timekeeper, etc., about the mine; one-half his time being charged to milling, the other half to the mining costs. At some works the mine superintendent performs these duties.

The power for a 40 stamp mill is calculated as follows:—

	Horse-power.
1 Rock-breaker	12
40 Stamps	66
16 Concentrators	8
8 Shaking-tables	2½
1 Clean-up pan	1½
1 Revolving barrel and batea	2
Total	92

Ninety horse-power will suffice if the revolving barrel, clean-up pan, and batea are run when the rock-breaker is thrown out of operation.

The use of water-power, when practicable, effects a saving in cost of plant, labour, fuel (the price of wood in California ranging from 12s. 6d. to £1 0s. 10d. per cord, and $\frac{1}{15}$ to $\frac{1}{8}$ of a cord being consumed per ton of ore crushed), repairs, and lubricants, decreases the liability of fire, affords a ready means of extinguishing conflagrations (reducing the insurance premium), and is more constant than steam. Where sufficient head is available, especially in dealing with a limited volume of water, hurdy-gurdy wheels, such as the Pelton, which develops 75 to 80 per cent. of the theoretical power of the water, and under the most favourable conditions even several per cent. higher efficiency, are to be recommended. For low pressures Leffels turbines may be advantageously adopted.

Copper-plates.—Nothing in milling practice gives greater trouble than keeping copper-plates free from yellow stain—electro-plating is beneficial; and another way out of the difficulty is to substitute Muntz metal for copper as a material for the plates, providing the ore is not excessively acid or of very high grade.

Muntz metal is an alloy of three parts of copper and two parts of zinc, a small proportion (less than $\frac{1}{100}$th) of lead being commonly added to it. Plates of this kind can be kept in good condition by using a weak solution of sulphuric acid, and are supposed to set up a slight galvanic action,

which keeps the mercurial surface in good condition. If Muntz metal be unprocurable all one need do is to take a metal that is positive to mercury, such as iron, and attach a strip of it in contact with the copper, at the head and down each side of the plate. This forms a weak galvanic couple, and in this way the oxidation being transferred to the more positive metal, the amalgamated surface of the copper will be protected from any acidity in the ore. In preparing copper-plates, advantage is often found in coating them at the start with silver, or better still, gold-amalgam, and the electro-plated copper-plates before alluded to are frequently used. There are different methods of amalgamating ordinary copper-plates, and the matter is of importance, as it has been found that when prepared with nitric acid they seem most liable to tarnish. A plate prepared with cyanide of potassium remains bright longer, whilst one which has been brought into condition with zinc-amalgam appears to stand better still.

The best results seem, however, to be obtained by using mercury in which a little cadmium has been dissolved, which can be advantageously used in small quantities both on the apron and in the mortar.* The vibrating amalgamated copper-plates, of which mention has been made to which an adjustable pitch can be given, mark a decided advance in battery amalgamation.

Copper-plates for amalgamating should not have been hard-rolled. To make them properly porous they ought to be annealed before use, heating the whole surface uniformly, and taking care that it does not get oxidized. They should weigh not less than 3 lbs. per square foot, and, within reasonable limits, the heavier they are the better. When laid smooth and true, the first dressing may be applied, by first scouring well with sand and wood-ashes, or they may be rubbed with fine emery powder, and washed with soda to make them perfectly clean and bright. After this, wash with clean water, and rub over with a solution of $\frac{1}{2}$ ounce of cyanide to a pint of water. Follow with a second washing of water applied warm, to remove the excess of cyanide, and rub on a mixture of fine sand and powdered sal-ammoniac, together with a little mercury, with a piece of blanket or a hard brush. Sprinkle the plate over with mercury, and allow it to remain on half an hour, then wash off the sand.

* In some cases it is found beneficial to add about $4\frac{1}{2}$ inches of a stick of sodium, $\frac{1}{2}$ an inch square, to a flask of mercury to quicken it, but it must be done gradually in an open dish, holding the sodium in a cleft stick, as the action is very violent. The exact quantity must be found by experiment, for too much will cause great loss of mercury and gold.

ain with the cyanide solution, and add as much more plate can absorb. Silver-amalgam can be best laid on ammoniac, using a piece of rubber belting.

tars, low discharge, and excessive quicksilver feed (which amalgam) decrease the percentage of the amalgam caught ixes.

ayer of hard amalgam should not be allowed to accumulate manently. To remove it, the plates may be occasionally iling water until it is sufficiently softened to be easily is is better than the ordinary method of sweating them.

: of the outside battery-plates and aprons of a 20 stamp ng for 18 months on £3 15s. ore (notwithstanding the tes were daily and carefully cleaned), has been known to rth of amalgam.

re only the "tenderfoot," or the "new chum," who old plates, much less accept the philanthropic offer of old ones (unless he had first previously sweated them a it is doubtful that he would be the gainer), unless they worn out.

ve formed by reducing the pitch of the fixed plates will a beneficial effect and save gold-amalgam, which would . This was proved by experience at the Spanish mine. take to follow up the aprons with narrow-plated sluice- bject of catching fine gold.. It would seem more rational elocity of the current by spreading it over a wider surface a. In Australia, it is a common practice to arrange the vith a fall of 2 or 3 inches at the head of each, the pulp falling vertically on to the coppers through an iron grating made slightly convex, which is punched with $\frac{1}{4}$ inch holes, and is supported across the head of the apron, so as to catch the discharge from the step above. The amalgam collects in thicker ridges below this grating, than on the lower surface of the plate.

No one who has been through different gold-mining camps can escape noting the fact that a modern stamp mill fully equipped with improved labour-saving appliances, and arranged so as to give the maximum of efficiency and economy, is comparatively speaking a rarity. There is more difference between the work done and results obtained, from a badly constructed and carelessly arranged stamp mill and those from a model battery, than there is between the latter and some other perhaps more efficient type of reduction process.

Day by day, milling becomes more systematically managed and scientifically worked. There is a tendency towards an increased use of wire-gauze screens in wet-crushing mills in some parts of California. They ought to give a higher duty, as the area of the openings is larger in proportion than in punched plates, and they show no tendency, as was formerly thought, to amalgamate the brass wire, which would destroy and choke them rapidly.

The necessity has been pointed out elsewhere of determining how much gold is being lost in the tailings, and an examination should be made as to the manner and cause of loss, with a view to seek a remedy.

The method of conducting such an investigation is as follows:—

1. Obtain a fair average of the daily tailings sample. Pan this down carefully to ascertain if free-gold, amalgam, or mercury is escaping. If there is an apparent loss the man in charge should be either replaced or instructed in his business.
2. A quantity of the average sample is next sized by passing it through, say 60 and 100 mesh wire screens, and each of the three sizes resulting, is weighed to determine the relative proportion of each, and assayed separately. If the sample which does not pass the 60 mesh screen, assays appreciably more than the finer sizes, the loss is evidently through enclosure in the particles of gangue; in other words through the ore not being reduced fine enough; and may be owing either to the free-gold or sulphides present. The sample should therefore be pulverized finer in a mortar to see whether it is to be ascribed to the one or to the other possible cause. The result will show whether finer crushing is likely to be productive of gain by amalgamation. If the assays of the three samples are nearly alike, finer crushing would clearly be disadvantageous, reducing the capacity of the batteries and promoting liability of loss. If the pulp passing the 100 mesh sieve goes highest, a coarser screen may be tried as likely to give increased capacity without extra loss.
3. A weighed portion of the original sample should next be panned-off to determine the percentage of sulphides present. The sulphides collected are then assayed to determine their value for subsequent treatment. In this connexion, test 1 must have been previously made, as any amalgam would, if present, lead to an entirely erroneous conclusion in estimating the value of the concentrates collected.

4. Note the loss of fine-gold in the slimes, as shown by the assay of the ore screened in the second test, and determine as nearly as possible by careful panning and assay of the concentrates after amalgamating the free-gold by hand, what proportion of the loss is due to the combined and what to the free-gold.
5. If the loss is high both in the slimes and coarser sizes, the gold is probably in a condition not susceptible to amalgamation, and if after trying the remedies that will be presently pointed out, no better results ensue, a small pan test may be made in the laboratory. In searching in this direction for a more perfect result the extra profit must, however, be carefully weighed against the additional cost.
6. Assays and analyses of the gangue and its component minerals, and a microscopical examination of the actual character of the gold, may in some cases give valuable assistance in enquiries of this nature.

The causes of loss have been stated in the earlier part of this paper as being due to (1) floatation; (2) enclosure in particles of gangue; (3) inaptness of the gold to amalgamate; (4) impure mercury; (5) bad condition of the plates; and (6) by its escape with mercury.

The loss from the first cause will generally increase, as pointed out by Mr. C. H. Aaron in the *New York Mining Journal*, August 10th, 1889, (*a*) with the fineness of the gold; (*b*) in absolute quantity, though not in percentage, with the richness of the rock in gold of this latter description; (*c*) in percentage, but not in absolute quantity, with the poverty of the rock in fine-gold; (*d*) with the quantity of water used; (*e*) with the muddiness of the water, hence a medium must be found in this respect so as to dilute the pulp as far as possible without running the danger of sweeping it away with too strong a current, (it is particularly in this connexion that shaking plates are of service); and (*f*) on the degree to which coarse particles of gold and amalgam are abraded and comminuted by the stamps, which is a strong argument against amalgamating as much as possible in the mortars using fine screens, with high discharge, and a minimum of water in the battery, except under special circumstances that justify it.

The loss from the second cause (enclosure in the particles of gangue) can only be remedied by finer crushing. There is a limit to this being done, however, as the finer the stone is crushed the finer will the particles of gold be reduced as well as the rock, the smaller will be the output of the battery, and the gold obstructed by the fine particles of gangue with which the water is surcharged, will have less opportunity of settling and

amalgamating. In such a case the rock may be submitted to two distinct operations of crushing and amalgamation with an intermediate separation of the slimes, but this entails extra expense which must be set against the additional saving effected.

At the Plumas Eureka mill, the tailings of the battery, as a case in point, are taken up by Italians (who pay a royalty for the privilege), and passed into wide shallow sluices which retain the sand while the slimes escape. The sands are then ground and amalgamated in arrastras.

Inaptness of the gold to amalgamate, the third cause of loss, may be overcome chemically or mechanically. In the latter case, by passing it through a machine such as a pan or mill, in which it is ground and brightened, or by warming the battery water if the inertness of the amalgamation be due to cold. If, however, this difficulty is of a chemical nature, advantage may be found in allowing a small stream of potassium cyanide to trickle from a tank into the battery, but care must be exercised in its use, or a fresh source of loss, from gold being carried off in solution, is liable to arise.

A little red oxide of mercury dissolved in the potassium cyanide will cause every particle of free gold exposed, to be instantly coated with quicksilver, and is efficacious where other means fail, but the use of such reagents is costly.

The precipitation of lead or copper in the mortars may be prevented by adding soda or milk of lime to the water used, or by causing it to flow over broken limestone, if the difficulty be merely in the water itself.

As regards impure mercury, the fourth cause of loss, the presence of lead, copper, mercurous oxide, sulphur, etc., is admitted to be injurious. Sometimes an ore contains sulphate of lead, which being to some extent reduced by the iron of the battery, will amalgamate with the quicksilver. In a similar manner soluble salts of copper either in the ore or the water cause a precipitation and amalgamation of the copper, which, though less harmful than lead, is still injurious. In all such cases the quicksilver after being strained should be purified before re-use.

To remove small quantities of copper and lead from mercury, retorting is not necessary. Such impurities may be eliminated by keeping the mercury for some hours in an enamelled pot, under dilute nitric acid (which is best warmed), and occasionally the mercury should be stirred. The acid will dissolve the copper and lead until it becomes saturated. It may also dissolve some mercury, but this will be deposited again when a fresh lot is treated, or it can be recovered by immersing a strip of copper in the liquid.

As the amalgam removed from the plates is liable to contain a little copper, it is well to always keep it in stock under acid. When wanted for use it should be washed with clean water. Oxygen, sulphur, and chlorine may be removed from quicksilver by adding a little sodium amalgam, avoiding an excess. Lead is not wholly removed by retorting, unless the quicksilver be covered to a depth of an inch or two with powdered charcoal.

If a solution of potassium cyanide in which red oxide of mercury has been dissolved is used in the pans (as is sometimes done, with the object of facilitating amalgamation), a little zinc-amalgam should be added to the pan towards the close of the operation, to avoid loss of gold in solution.

With regard to the fifth cause of loss, Mr. E. B. C. Hambley, in trials made at one of the Indian mills, in 1886, by substituting efficient for inferior plates, and looking after them properly, succeeded in saving 63·23 per cent. of the total gold caught, as compared with 33·33 per cent. yielded previously.

When ore is crushed dry there is great danger of loss, owing to fine particles failing to become thoroughly wetted, in consequence of which (although specifically heavier) they will often float for a long time on the surface of the water.

The sixth cause of loss, that of quicksilver carrying gold, may often be caused in the same way by flowering, owing to its attrition in water with sand, powdering it to an almost impalpable dust. The loss in itself may not be a serious item, but when charged with gold it is one of some moment, as it has been proved at the North Star mill, Nevada County, that £10 8s. 5d. worth of amalgam escapes the aprons and sluices which would be lost but for the shaking-aprons and concentrators below, intervening between the tailings-race and the battery.

Another loss of mercury occurs in handling, and a third in retorting the amalgam, a portion remaining unexpelled from the bullion, and some escaping condensation. At the Keystone mill, 10 ounces per month are lost in this way. Minor sources of loss are oxidation, sublimation (extremely slight at ordinary temperatures), combination with base metals in the ore, and adhesion to metallic particles in the gangue.

What is true in regard to not cleaning the plates too frequently does not apply to the battery, as it is generally beneficial, more especially if the ore contains coarse gold, which is caught in the box, to clean it out every two or three days, notwithstanding the loss of time.

The common practice of running for a month without a clean-up of the boxes, is in such cases bad, as the amalgam runs a great chance of being flowered and thrown out of the mortar, resulting in loss. Steel tappets and

cams are coming into general use, and it is an advantage to counterbore them. Split cams are convenient for changing if a breakage occurs.

Though the hardness of quartz as tested by scratching is nearly uniform, the facility with which it can be crushed depends on its texture, *i.e.*, its friability or compactness. The smaller the rock is broken, to a uniform size before it goes to the battery, the lower and more uniform will be the feeding, admitting of lower and more rapid drop. Hence more ore can be crushed for the expenditure of a given amount of power.

Battery Water.—The amount of water fed to the battery depends on the number of stamps in each battery, character of the ore, size of screen, and discharge of the mortar. Clayey and highly mineralized ores require a maximum amount, and a roomy, long or double-discharge mortar, more than a narrow single-discharge type. The quantity used per ton of ore stamped in California runs from 1,000 to 2,400 gallons. The mean amount is about 1,800 gallons. From ¾ to 1½ miner's inches per battery should be provided; a miner's inch is about 16,800 gallons supplied per 24 hours.

Practice in Colorado.

The following interesting particulars, chiefly taken from a series of papers on variations in the milling of gold ores by Mr. T. A. Rickard, which appeared in the *New York Mining Journal* in 1892 and 1893, illustrate the differences that obtain in milling practice in Colorado, Australia, and New Zealand.

At the Hidden Treasure mill, the stamps of the old batteries are furnished with screw tappets which have given place in the newer ones to the gib-and-key method of attachment. The stamps weigh 550 lbs., fall 30 to 32 times per minute, and drop in order 1, 5, 2, 4, 3. Each stamp makes from 1¼ to 1½ revolutions at each fall, depending on the amount of grease in the cam shaft. The discharge, measured from the top of the die to the bottom of the screen, is 13 inches when new dies have just been put in, increasing as they wear down to a maximum of 15 to 15½ inches.

The shoes are 5½ inches deep, and 8 inches in diameter. The dies are plain and cylindrical, fitting into a round seat in the mortar bed. They are 3½ inches deep, and slightly wider than the shoes, and are kept in place by being tightly packed round with tailings. The shoes weigh 83 to 86 lbs. each, the dies from 46 to 48 lbs., both being made of locally manufactured cast iron. The wear of the shoes is 11·3 ounces of iron per ton of ore crushed, that of the dies 4·5 ounces. At present fifty heads are employed on custom work, and these crush faster than the twenty-five fed with mill-rock from the California mine.

COMPARATIVE TABLE OF GILPIN COUNTY MILLS.

Name of Mill.	Hidden Treasure.	Gregory Bobtail.	Randolph.	New York.	Prize.
Number of stamps ...	75	125	50	75	25
Weight of each stamp(lbs.)	550	550	500	600	500
Number of drops per minute	30 to 32	27 to 30	30	26	28 to 30
Height of drop (inches) ...	16 „ 18	16 „ 18	16 to 18	18 to 20	15 „ 17
Depth of discharge at issue (inches)	13 „ 15	11 „ 13	14 „ 16	13 „ 15	13 „ 16
Capacity per stamp head (tons)*	1·14	1·04	·93	1·07	0·80
Capacity of entire mill (tons)	85	130	48	80	20
Size of screen (No.) ...	1½	1 & 2	1½	1½	1½
Description of screen ...		Burr slot	alternate	punched.	
Percentage of concentrates per ton of ore	13	14	20	15	12
Value of concentrates per ton	£3 2s. 6d.	£2 1s. 8d. to £5 4s. 2d.	£2 1s. 8d.	£1 9s. 2d. to £2 1s. 8d.	£2 1s. 8d. to £5 4s. 2d.
Percentage of bullion obtained in retorting ...	40	40	33 to 47	40	35
Fineness of bullion (per 1,000)	782 to 786	800 to 850	750 „ 850	750 to 800	750 to 775
Life of the screens (days)	81	60	16	25	75
Loss of mercury per ton of ore (dwts.)†	4·3	5·2	9·8	3·7	9·7
Consumption of water per stamp per minute (gallons)	2	2·3	1·4	1·3	1·5

Custom milling is a great feature of the mills of this section. The charges are £1 11s. 3d. per day per battery of five heads, including the concentration of the pyrites and their shipment on the railway cars. For small lots the rates are £5 4s. 2d. per cord for milling and 8s. 4d. per cord for concentrating. This last charge varies from 4s. 2d. to 12s. 6d. according to the percentage of pyrites present.

The cord of mill-rock above alluded to is equal to 7½ to 8 tons, while one of smelting ore averages 9 to 10 tons. The screen is of the burr-slot description, the slots being horizontal and alternating. No. 1½, equal to a 50 mesh wire screen, is usually employed. The screen surface is 4½ feet by 8 inches, 200 feet of screens being used up in a year equivalent to 66·6 screens annually. The average life of a screen is therefore 81 days. With the ore from the California mine they last three months by turning the lower portion, which wears most rapidly, to the top. The amalgam

* 2,000 lbs. each.

† Mercury is sold by avoirdupois weight—a bottle contains 76½ lbs.

yields 30 to 50 per cent. of bullion, which contains in addition to the gold, 207 to 211 thousandths of silver. Six and a half bottles of quicksilver are lost annually, representing 4·3 dwts. per ton of ore crushed by 75 stamps.

Most of the gold is caught in the mortar-box, which is effected to a slight extent by the free mercury added, but chiefly by two amalgamated plates arranged along the front and back of the box. They are both of plain copper, both 4½ feet long, the back one being however 12 inches wide, the front one 6 inches. The former is set at an angle of 40 degs., the latter nearly vertical.

The front of the battery above the screen frame is covered with canvas, by lifting-which the mill man can introduce his arm, and tell by the feel of the front-plate whether the right quantity of mercury has been added by the feeder. This saves stoppage of the battery and removal of the screen. On the average, the feeder adds half a thimbleful of quicksilver every hour. A test has shown that in crushing 8 tons of ore, carrying ½ ounce of gold per ton, 4½ ounces of mercury were added. After the first six hours a drop as large as a medium sized pea added every hour sufficed.

The gold-saving appliances following the battery are amalgamating tables, blankets, and concentrators. The first of these are 12 feet by 4 feet in one length, covered with copper, with a grade of 2⅛ inches per foot, which is greater than in California.*

In crushing 24 tons of ore carrying ½ ounce per ton, it was found that one such table required 5 ounces of mercury to dress it, whilst 3 ounces were used in dressing the front inside-plate, and 4 ounces for the back one.

The blanket-strakes or strips are 3 feet long and 18 inches wide. They are washed three times per 24 hours and serve to arrest any escaping amalgam mercury, rusty gold and the heaviest pyrites, together with particles of stone to which gold is still attached. Unless to save this last class of material, concentrators could replace them without loss. From the blankets the pulp passes to concentrators known locally as bumpers, a variation of the Rittinger.

In the Hidden Treasure mill there are 5 of these tables. The speed is regulated by the percentage of pyrites present, averaging 130 strokes per minute. Of the total amalgam obtained, ⅔ is yielded by the inside-plates. On cleaning-up, the sand found in the battery round the dies is not panned but returned to the mortar. The outer tables are cleaned-up every 24 hours, but the inside-plates only every 48 hours. With poor

* Page 238.

ore the last named period is prolonged. At the general clean-up the amalgam from the plates is placed in a mortar and ground with hot water till of even consistency, the dirty water being decanted and mercury afterwards added to thin the amalgam.

Thus prepared it is run from one porcelain dish to another several times, and as the dirt and pyrites rise to the surface they are skimmed off by hand. The clean amalgam remaining is pressed through canvas. The skimmings resulting are re-introduced into the mortar and re-ground with fresh mercury and hot water. When fairly clean, a bit or two of potassium cyanide is added to render the mercury more lively.

In retorting, the retort is either chalked or lined with paper. The balls of dry squeezed amalgam are introduced into the retort, broken with an iron rod, and pressed down till hard and uniform. The cover is then put on and luted down with clay. Fine pan-sludge is often used in Australia, and forms a good lute. The only chemical used is potassium cyanide. Of this twenty-six 10 lbs. canisters were used in a year in the treatment of 28,793 tons of ore crushed. The tables are dressed every 12 hours with a weak solution containing 2 ounces dissolved in 3 gallons of water. The tables are brushed twice daily with a mop, mercury being sprinkled over them if the amalgam be too dry. The feeding is done by hand, and there are no rock-breakers. Feeders are paid 12s. 6d. per shift, and there is one for every 25 stamps. The mill is worked four months in the year by water power, four months by steam, and four months by both combined.

Firewood costs 19s. 9½d. per cord delivered. The cost of milling in 1890 was at the rate of 3s. 6d., but in 1891 it was decreased to 3s. 3d. per ton. The labour employed is thus distributed per month of 30 working days, 75 heads. One mill-man £36 9s. 2d., 1 assistant £20 16s. 8d., 6 feeders (3 per shift) at 12s. 6d. per day, £112 10s. per month, 2 concentrator men (1 per shift) at 12s. 6d. per day, £37 10s., total £207 5s. 10d., or 1s. 7d. per ton. Twelve hour shifts are worked.

While all the mills of the district are much of the same type and are engaged in crushing ore of much the same character, there are slight differences of detail in them. The stamps are all of light weight, rendered necessary by their high drop, which would be impossible with stamps of 850 to 900 lbs. The speed is also directly affected by the same cause, for the work required to lift the heads to a height of 16 to 18 inches prevents the rate of fall exceeding 32 drops per minute for good work, 40 is probably the practical limit, but the present tendency is in favour of keeping

it up to a speed of 32 drops, rather than running slower. The New York mill has the longest drop, but in this respect it follows the older practice.

The Gregory Bobtail mill fairly indicates the construction of the most recent plants in regard to depth of discharge. The precentage of concentrates obtained from the ores of this district ranges from 12 to 20 per cent., showing the refractory character of the mill-stuff. The value is very low, averaging £2 1s. 8d. to £2 10s. net per ton. The concentrates and blanketings undergo no further treatment, and are shipped direct on railway cars from the doors of the mills to the smelters at Denver. The freight is 6s. 3d. per ton, and cheap concentration supplemented by light smelting charges, alone makes the treatment of the concentrates profitable. Ninety-five per cent. of the silver and gold contents as fixed by assay is paid for, £1 13s. 4d. being deducted per ton for treatment; formerly a minimum rate of £1 10s. 2½d. was allowed, when this class of material was less plentiful. The retort percentage depends on the coarseness of the gold and the thoroughness with which the amalgam is squeezed, imperfect manipulation causing a difference of as much as 10 per cent.

The small capacity of the mills, the slight use made of the blankets, and the high slope of the amalgamating-tables (1½ to 2½ inches per foot), considering the deep discharge, account for the comparatively small amount of water used, viz.: 1½ to 2½ gallons per minute per stamp. The screens are of local manufacture, and are made of planished iron (an imitation of Russian sheet), size No. 24. The openings are straight slots set alternately. Nos. 1, 1½, and 2 most in use, are considered equal to 60, 50, and 40 wire mesh; they have nothing like the same discharge-surface however; a large proportion of the pulp being kept in the box till it would pass a 100 mesh wire screen. Their use is only justified by the main idea of Gilpin County practice, that of retaining the pulp inside the box till it is amalgamated. The side which carries the burr edge of the punched openings is always placed facing inside, to break the pulp and prevent choking.

There is a wide difference, as will be seen from the table, in the life of the screens in the different mills—far greater than can be explained by the greater or less attention of the mill-man, and the extent to which he is willing to allow the screen-slots to be enlarged by wear. Mr. Rickard accounts for this by their order of succession of the mills along the creek which supplies them with water. The Hidden Treasure mill receives water comparatively clean, and after having used it returns it to the

creek with a certain percentage of sulphuric acid, derived from the contact of the water with the pyrites. It is not surprising therefore that its screens last the longest—81 days. The Prize mill, in the same way, adds its quota of sulphates to the water, to the additional injury of the Gregory Bobtail screens, which only last 60 days.

The lives of the screens of the New York and Randolph mills are measured by days (25 to 16) instead of weeks, owing no doubt to the additionally corrosive action of the water lower down, due to the acid waters which issue from the underground workings of the Gregory mine above them. Taking the Hidden Treasure as most truly typical of the life of the screens in this district, we find 432 tons of ore pass through a grating before it is worn out. At Grass Valley, a screen will live to pass 200 tons, and at Bendigo (Australia) 134 tons. The very roomy character of the Colorado mortar-box probably accounts in part for this favourable comparison.

The loss of mercury is greatest in the mills with the deepest issue, which bears out what has been said elsewhere as to the danger of stamping it to flour, *i.e.*, subdividing it into minute globules, which are liable to become coated with pyrites or foreign matter, and are then borne away by the water. The variable loss of mercury at different mills may also be largely explained by the larger number of clean-ups where more lots of ore are treated coming from different mines, which lead to additional manipulative losses.

The ore of the California mine, treated at the Hidden Treasure mill, is representative of the ore of the district. It consists roughly of 15 to 20 per cent. of quartz, and 60 to 70 per cent. of vein-filling (other than quartz), an altered form of the rocks enclosing the vein. As these latter consist of gneiss, alternating with granite and mica-schist, the gangue is largely felspathic.

Of the metallic constituents of the ore, iron and copper pyrites predominate; grey copper (fahlerz or tetrahedrite) arsenical pyrites (mispickel) and galena are also present in noteworthy proportions. Blende is sometimes seen and chalybite (carbonate of iron) appears occasionally. The grey copper, which is here antimonial, is generally remarkably favourable for the presence of gold, a fact which would prove a valuable index in selecting the ore, were it not so often confounded with arsenical pyrites. Quartz (especially favourable when of a blue tint) is always associated with the pyrites in rich ore. The writer may here remark that he has observed this same peculiarity of colour in the rich quartz of the Mysore mines in India.

The following results of assays made by Mr. Rickard, upon a typical piece of ore broken in the 1,700 feet level of the California mine, throws some light on the distribution of the gold and silver :—

Mineral.	Gold. Ozs. per Ton.	Silver. Ozs. per Ton.	Remarks.
Iron pyrites ...	0·65 ...	4·85 ...	White coarsely crystalline.
Copper pyrites ...	0·85 ...	53·50 ...	Flaky dark yellow.
Grey copper ...	0·90 ...	38·65 ...	Chiefly covering the last.
Blende	0·16 ...	6·45 ...	Black crystalline.
White quartz ...	3·32 ...	7·35 ...	Opaque massive, with small crystals of pyrites throughout.
Bluish quartz ...	3·56 ...	5·84 ...	
Flinty quartz ...	0·18 ...	1·90 ...	Brown vitreous.
Felspathic gangue	0·90 ...	2·35 ...	Soft granular white.

This analysis bears out the experience of the mills, half the gold-contents of the being are extracted by the first amalgamation in the mortar box. The gold cannot therefore be chemically combined with the pyrites. On the other hand, the more highly mineralized the ore the richer it usually is also. There is no doubt the silver-contents are for the most part associated with the copper-bearing minerals, while the gold is enclosed in the quartz, especially when that quartz is immediately associated with pyrites. Neither blende nor galena is an attendant upon the gold, and both are a nuisance in the mill.

The following figures, representative of the output for 1890, further illustrate the nature of the ore—150·44 tons of smelting ore averaging £19 6s. 7½d. per ton net; 1,376·03 tons of concentrates averaging £3 2s. 9d. per ton net; 10,320·57 tons of mill-stuff averaging £1 10s. 11d. per ton.

The smelting ore is the high grade sulphide ore picked out at the mine and shipped direct to Denver. The mill-stuff yielded 4,766·39 ounces of bullion, worth £3 9s. 4½d. per ounce. Of the total tonnage, 90 per cent. was mill-ore, representing 84 per cent. of the total value. The mill-ore yielded 13 per cent. of concentrates.

A test made in March, 1891, to determine the completeness of the mill extraction on a lot of 8,400 lbs. of ore, which contained 4 per cent. of moisture (leaving 8,064 lbs. net) showed, after passing through a breaker and rolls (specially erected for accurate sampling), an assay value in gold of 1·85 ounces, and in silver of 8·75 ounces per ton.

The contents of the 8,064 lbs. were therefore 7·46 ounces of gold and 32·86 ounces of silver. At the smelter such ore would be worth £5 12s. 10d. per ton net (smelting charges being £2 10s. per ton and 95 per cent. of the gold and silver being returned and paid for on New York quotations) or a total value of £22 14s. 11d.

This ore was sent to the mill and yielded, after treatment for 18 hours in a 5 stamp battery, 6·70 ounces of bullion, worth £3 6s. 8d. per ounce, or £22 6s. 8d. and 2,325 lbs. of concentrates containing 15 per cent. moisture, leaving 1,977 lbs. net equivalent to 24½ per cent. of sulphides in the sample. The assay of the concentrates gave 1·76 ounces of gold and 10·37 ounces of silver per ton, or a total for the 1,977 lbs. of the above assay value, representing £9 3s. 5½d., or deducting smelting charges, £7 10s. 1½d. net.

To compare these results, the milling cost was at the rate of 3s. 6d. per ton; therefore the mill returned, after making all deductions, £29 2s. 8½d., estimated as follows:—Bullion, £22 6s. 8d.; concentrates, £7 10s. 1½d.; total, £29 16s. 9½d. Deducting milling cost at 3s. 6d. per ton 14s. 1d. equal £29 2s. 8½d.

At the smelter, the amount received, owing to the larger deductions and charges, reaches the smaller sum of £22 14s. 11d. Commercially, therefore, the mill ore paid the miner better than the smelting ore.

As a test of the mill-work the figures are as follows:—There was in the ore, 7·46 ounces of gold and 32·86 ounces of silver. There was extracted as bullion, 5·25 ounces of gold and 14 ounces of silver, and in the concentrates, 1·74 ounces of gold and 10·22 ounces of silver, or a total of 6·99 ounces of gold and 24·22 ounces of silver. Thus the mill, including the value of the concentrates, saved 93·8 per cent. of the gold and 74 per cent. of the silver.

The mill did not, however, complete the extraction of the gold and silver in the concentrates, so that it actually obtained, by amalgamation alone, 70·4 per cent. of the gold, and 42·6 per cent. of the silver.

This is said to be fairly representative of returns obtained on a large scale. Generally speaking, it has been found that the mill yields as many ounces of base bullion as there are ounces of pure gold found in the ore by fire-assay. The mill gold is 780 fine.

Considering that Gilpin County ore is probably one of the ores running highest in sulphides treated by amalgamation at the present day, the extraction of the stamp mills is certainly extremely good. That this is so, is due to the proper recognition of the necessity for altering modes of treatment in accordance with differences in the character of the ore treated—the first principle of successful milling.

The ore of the district has been described, and it may be added that in this locality ordinary panning, except with surface ores, will give no colours of gold, even with material which in the mill yields rich returns.

The *raison d'être* for the roomy mortars, slow drop, and deep discharge which characterize the Black Hawk mills are fully explained by Mr. Rickard. His views entirely agree with the writer's, expressed in a later part of this paper.

The absence of rock-breakers in the Gilpin County mills is largely due to their bad situation, which was chosen to utilize as far as possible the motive power of the creek. The position of the mill-buildings in fact (except at the expense of elevating the ore) prevents the erection of ore-bins and grizzleys, which are necessary adjuncts for running a stone-breaker economically.

Though the mills, crushing as they do very slowly, have not the same crying need for a rock-breaker as that which exists in a Californian or Australian mill, it is no doubt a defect; for apart from the improvement in the feeding which follows the introduction of a breaker, the irregular work of the sledge must tend largely to increase the strain on the mill machinery, as evidenced by the wear and tear of the shoes and dies which in this section is excessive.

On the score of regular and accurate feeding, the automatic machine is preferable to the average man, who cannot always resist the temptation of an occasional pipe or spell of rest. Their economy is most apparent of course where stamps crush fast, but even in a case like that of Gilpin County practice, an advantage may be shown in adopting them, notwithstanding that one man feeds 25 heads.

For 75 stamps, the cost of feeding comes to a total of £1,354 3s. 4d. per annum, while, on the other hand, if a mill of the kind were supplied with the most expensive of self-feeders, the cost of the additional plant would not exceed £834 6s. 8d.

Feeding machines are, however, of little use unless preceded by grizzleys, breakers, and ore-bins, and therefore recognizing the unfortunate position of the mill-buildings at Black Hawk (chosen in the days preceding the introduction of improved labour-saving appliances), however averse one may be to methods out of date, and machinery which is incomplete, one cannot say that, from a shareholder's or mill-man's point of view, present practice could be advantageously altered.

The Black Hawk mill-man has been trained in the best school, that of experience, supplemented by the necessities of keeping a close watch over the treatment of an ore subject to frequent changes in mineralogical con-

stitution. The competition for customs ores renders him in fact careful in the treatment of the stone, and keenly awake to any possible improvement of method promising ultimate economy.

The milling is recognized to be as important as the mining, and mills are not placed under the direction of men who are simply good miners, good chemists, or anything you will, but assuredly bad and inexperienced mill-men. In Australia it is not uncommon to consign a first-class battery to the tender mercies of an engine-driver who, in addition to tending the engines, is supposed to supervise the general mill work.

In Gilpin County, the management of the mill claims an equal or greater share with that of the mine. The needs of the district have produced men who are fully conversant with the bed-rock principles of gold milling, and such men are not too well paid, though earning more than the mine foremen.

Customs milling has had the beneficial effect (by rendering the mill owners anxious to gain the confidence of the public) of placing the right sort of men in charge, and by encouraging competition in doing good work has benefited the mill-men themselves. Gilpin County practice in fact leaves the impression of good work, intelligently and conscientiously done—two factors of the first importance to the mine owner and mining industry.

Colorado's production of gold appears to have been largest in 1886, when it is stated to have produced 4,450,000 dollars (£1,250,000) worth as against 3,883,859 dollars (£809,135) in 1889, whilst its silver output was valued at 23,757,751 dollars (£4,949,530) in 1889.

Practice in the Thames District, New Zealand.

This once famous mining district, also known by the Maori name of Hauraki, is situated in the north-eastern corner of New Zealand. Though the output has now dwindled to about 30,000 ounces per annum, this has been in its day one of the richest gold-fields of the world. In 1871, the output was 330,326 ounces valued at £1,188,728. The Caledonia mine in the first 12 months' operations produced 10 tons of gold and paid £600,000 in dividends. The maximum depth yet attained is 600 to 700 feet, and the veins have unfortunately proved far less rich in depth than near the surface. The development of the gold-field has been crippled by share-jobbing, the curse of many other Australasian camps. Few districts have had so brief but brilliant a record, and few perhaps have lost such a large proportion of the gold extracted from the mines. Milling is conducted under the difficulties presented by ores of a very complex

composition, but so far the efforts made to overcome them have been of a very elementary description. It is for this reason that the tailings-mills on the field are to-day amongst the most profitable undertakings.

Briefly stated, the method of milling consists in catching all the free gold by means quite unsuited to the character of the ore, and allowing the remainder to go to enrich the sea-beaches. The character of the ore and lode-formation in which it occurs, help partly to explain a state of things which calls for such severe criticism. The bulk of the gold comes from narrow veins and extremely rich pockets traversing a decomposed andesite. Such ore-bodies must necessarily be uncertain in behaviour and of limited extent. Like the deposits of Nagyag and Veraspotak in Transylvania, to which they have a striking resemblance (the country rock, the deposition of the ore, and the character of the gold specimens, as well as their alloyage with silver, being very similar), the pockets found on the Thames are occasionally of extraordinary richness. For instance, one lot of 2 tons 8 cwts. crushed 2½ ounces per pound, and a boulder of 2½ cwts. yielded 3,500 ounces. These crushings of small quantities of very rich ore pay the dividends, and the bulk of the output being of small value in proportion to the specimen ore, the former has been sacrificed to the latter.

The chief features of the milling are indicated by the subjoined table:—

Name of Mill.	Saxon.	Moanataeri.	Cambria.	Kuranui.	Comer.
Number of stamps	32	40	20	20	20
Weight of stamps (lbs.)	785	659	620	670	840
Number of drops per minute ...	72	66	76	70	63
Height of drop (inches)	9	8	9	5½	6
Average depth of discharge (ins.)	2½	2	3	2¼	2½
Capacity per head (tons) ...	1·8	1·4	1·7	2·5	3·6
Capacity of mill (tons)	58	55	35	50	72
Description of screen		Punched	Russian	iron.	
Number of holes per square inch	148	170	180	160	160
Fineness of bullion (per 1,000) .	663	641	674	605	589
Percentage in retorting	42	40	40	45	48
Life of screens (days)	6	6	5	5½	5
Loss of mercury per ton of ore (dwts.)	14·5	15·2	—	—	—
Number of berdans	8	21	15	7	5
Number of other pans	3	4	—	—	—

No concentrates are obtained in any of these mills.

The Saxon mill it will be observed contains 32 stamps, with an extra single stamp kept for the treatment of specimen ore. The weight of the stamps, shoes and dies, and rate of drop are not the same, though the latter (subject to variations) averages 72 per minute. The depth of discharge is about 1 inch at the time of putting in new dies, increasing to

a maximum of 4 inches as they wear down. The height of drop is from 8 to 11 inches, depending on the hardness of the ore. The single stamp weighs 7 cwt. It is given a 7 inches drop, and a speed of 60 drops per minute. This specimen stamp is a curious feature of all the mills. The screen or grating used is of Russian iron, imported from Swansea. The life of a grating averages 6 days. The openings are round, punched with 148 holes per square inch. The loss of mercury is at the rate of a bottle (75 lbs.) per month, three bottles being kept in stock. No mercury is used in the mortar save for the specimen stamp. It is employed on the plates and in the ripples or wells, and in the pans. Owing to the flowering produced by the pans, the loss of mercury is excessive, viz., 14½ dwts. per ton of ore treated. The amalgam retorts about 42 per cent. The ore brought to the mill is discharged into stalls behind the batteries. The stone is there spalled and hand-fed. The ore varies in hardness according as the andesitic vein-filling is decomposed. The quartz itself is often saccharoidal. The shoes and dies of local manufacture are of white hæmatite cast-iron, the die differing from the shoe in being unchilled. The former is 10 inches in diameter and 4 inches deep, the latter 9½ inches in diameter and 10 inches high. The die is cast with a flanged footing to keep it in position. The mortar-boxes are faulty in design, being too roomy inside.

The pulp is discharged on amalgamating-tables 7 feet long and 4½ feet wide. These are in three divisions, of which the two upper ones only are copper-plated. The first length is 2½ feet, inclusive of a well 2½ inches wide. This well contains mercury. The next division is 18 inches long. The ripples (riffles) are four in number, one only (that already mentioned) containing mercury. The other three (2 inches deep) are blind-ripples. The gold saving is effected by the plates and wells, and indirectly by blanket-strakes whose residues are treated in pans. The mortar-box is merely a crusher, not an amalgamating appliance.

The plates of Muntz metal are roughly cleaned every 4 hours. The wells are of little assistance, and are really unsuitable to ores containing a notable percentage of sulphides. They tend also to conceal the careless use of mercury.* The surface of the bath of mercury is constantly coated with a scum of sulphides, which prevents contact with any gold passing over it. The wells are skimmed with a cloth every 4 hours, and

* With plates immediately in front of the battery-screens a check is afforded of the work inside, and of the proper proportioning of the quicksilver feed or the reverse, while the well hides the fact, for a time, of a too liberal use of mercury or otherwise.

the mercury placed in them is squeezed once a week. The six wells, one to each battery, catch 12 ounces of amalgam, out of the weekly yield of 200 to 250 ounces from the entire mill.

The blind-ripples are cleaned with a scoop every half-hour and merely catch the heavier sand and sulphides which go to the pans. The blankets are washed every hour, the time, more or less, varying with the richness of the ore, and proportion of sulphides. This washing is done by one boy on each shift, of which there are three. Each boy is paid £1 per week. The blanketings go to the pans. The mill contains the following pans:—

	Diameter. Ft.	Ins.		Depth. Ft.	Ins.
8 Berdans	4	6	...	0	9
2 Watson and Denny	5	4	...	2	6
1 Price (local)	5	8	...	2	7

The berdans and one Watson and Denny pan treat the blanketings. The other two pans work tailings. The berdans have a pitch of 16 inches in 3 feet 6 inches, or 1 in 2⅜. The speed is 23 revolutions per minute. The amalgam is removed every 24 hours. A drag is used, which consists of a slipper or shoe weighing 196 lbs., and a top or boss of 233 lbs. weight. They are held together by a key. In Queensland two bolt-ends are cast into the shoe, and passing through the boss, are secured with nuts. The two parts are cemented together. A shoe lasts about 4½ months.

The distribution of the amalgam in the mill at the fortnightly clean-up preceding Mr. Rickard's visit was as follows:—Plates, 223 ounces; mercury wells, 24 ounces; pans, 43¾ ounces; 35 lbs. of specimen ore, 164 ounces; this gave 203 ounces of base-bullion, which yielded 200 ounces 4 dwts. of melted gold obtained from 476 ounces of amalgam, the balance being made up with skimmings. etc. The ore treated gives an average yield of 15 dwts. per ton.

The following figures show the cost of treatment at the Saxon mill per 24 hours with 33 stamps, crushing 63 tons, including the specimen stamp:—3 feeders at 6s. 8d. per shift of 8 hours, £1; 3 boy-feeders at 5s., 15s.; 3 blanket-boys at 3s. 4d., 10s.; 3 amalgamators at 8s., £1 4s.; total labour, £3 9s. The head amalgamator is manager of the mill; the cost of labour is therefore 2s. 10½d. per ton, while the total cost, including wear and tear, alterations to machinery, interest on capital, etc., is 4s. 1d. per ton. At the Moanataeri mill, which has a larger plant, the cost is 3s. 9d. per ton. The disparity of drop, shown in the table, is accounted for by the fact that the three first-named mills are treating ore coming from a greater depth than the others. Being harder, the stamps in the former

batteries have a fall of 8 to 9 inches as compared with 5 to 6 inches in the other two mills.*

The depth of discharge from the top of the die to the bottom of the grating is left to depend on the wearing down of the die. The importance of having a depth of discharge, suited to the particular mode of working aimed at and the particular ore treated, is a point quite unappreciated. The holes in the screens vary in number from 148 to 180 per square inch. Owing to the presence of acid sulphates, derived from the oxidation of sulphides, the life of the screens in the mills treating surface-ores is shorter than in those where harder stone is crushed. Owing to the greater speed of crushing, however, the screens in the Comer mill, for instance, last during the passage through them, of 90 tons, as against 54 tons at the Saxon mill. Their life, which is comparatively short, is due to acid mine waters, which act also, indirectly, on the partially decomposed metallic sulphides in the battery. This action of the mine waters of the district is so marked as to suggest the effects of a slowly dying solfataric action.

The loss of mercury is high (75 lbs. per month for 30 stamps), due to flowering, caused by grinding in the pans, and sickening produced by the presence of antimonial and arsenical minerals in the pulp. The bullion is of low grade, having a fineness which ranges from 589 to 674 per thousand.

The lodes, it has been said, are small and irregular; they traverse a hornblende-andesite, which is often brecciated. The lode-formation is confined to certain belts marked by the decomposition of the country rock, the mill-stuff carrying this latter in large proportion. The ore is silver as well as gold-bearing. The gangue is largely quartz, which penetrates the vein in veinlets and stringers. Sometimes the quartz is soft and saccharoidal. While the gold is frequently visible in the quartz in minute threads and particles, it is also largely associatied with copper and iron pyrites, blende, galena, etc. Silver occurs as pyrargyrite, proustite, argentite, etc. The two precious metals are found associated with tellurium as petzite and sylvanite; selenides are known to exist in the ore; and antimony, in beautiful crystals of stibnite, is often seen.

Generally speaking, the ore has a very variable hardness and composition; it contains a very large variety of metallic sulphides, and must be

* With a soft decomposed ore, or one in which the gold is very fine, a slow rate of drop is generally to be recommended. Soft and high sulphide ores should usually also be stamped with a low drop.

considered as a combined silver and gold ore, containing from ½ to 10 per cent. of sulphides with an average of 2 to 3 per cent.; it approaches the boundary-line dividing a free-milling from a refractory ore.

Reviewing the chief characteristics of the milling, we see that the feeding is done by hand and is very rough, being left to boys, instead of employing trained men. The former shirk breaking big pieces of rock, preferring to throw them into the feed-opening, where, if they stick, they are belted-in with a few blows of a sledge-hammer. The result of the absence of rock-breakers and automatic feeders in lieu of this bad hand-feeding is seen in the excessive wear of the shoes and dies, averaging 14·5 ounces for the shoe and 7·5 ounces for the die. The feeding is also too high, the batteries being kept choked with ore, which reduces their efficiency.*

The mortar-boxes are of the same pattern, whether employed for rapidly crushing soft material or the slower treatment of average ore. As no amalgamation is done inside, they are too wide. When the mortar is merely a pulverizer, the pulp should be expelled as soon as possible; too roomy a box promotes dead-stamping, keeping in ore, after it has been stamped fine enough to escape under ordinary circumstances. When amalgamation takes place inside, the case is different, but at the Thames, it would seem expedient to use narrower mortars.

The destruction of the gratings is particularly rapid, owing to the unusual quantity of proto-sulphates of iron, copper, manganese, and alumina present; and with wet mill-stuff which has lain long in the stopes, or on the surface, the action is very marked. As is usual in Colonial mills, the screens are fixed vertical (instead of having a slight forward inclination at the top), which tends to wear out the lower portion, faster than the upper, as well as to diminish the discharging capacity of the screen-surface; as the ore particles are not necessarily obliged to strike an opening, to escape. The water shot against an inclined screen from the inside, has also the advantage no doubt of carrying a certain quantity of material out, in running down the inclined screen-surface, unless burred screens are used.

* An experienced millman can tell by the sound of his stamps if they are running properly. Any particular stamp can be tested by holding the stem lightly between the fingers, about 18 inches above the mortar-box, and allowing the hand to be raised and lowered, by the rise and fall of the stamp. If the stamp is overfed it will fall with a dull short stroke; if underfed, with a long vibrating, ringing blow (like a hammer on an anvil); if properly fed, with the sharp well-defined crack you ought to expect.

The mercury wells only confirm the general experience that with ores containing an appreciable quantity of sulphides, they are of little practical value, not being even of a type (like those of Clunes), which compels the pulp to pass through the mercury-bath, so far as that end is capable of attainment. The blind-ripples occupy an amount of time which appears quite out of proportion to the quantity of sulphides and heavy sand which they collect.

The blankets are washed at too long intervals at most of the mills running on company ore, and therefore, tributers (who understand this) generally attend to that part of the business themselves, washing the blankets and skimming the wells every half hour. The blankets, which are 2 yards wide cost 12s. per yard; they last three months.

The substitution of a drag, or pair of drags, certainly is an improvement on the ball, formerly used in the berdans, which wallowing about in the mercury and amalgam (at the bottom of the pan), presented extra chances of flowering them. The drag being fixed to the side of the pan keeps the grinding and amalgamation to some extent apart, but more or less mercury is carried round constantly with the sulphides by centrifugal action, hence the evil is but partly mitigated.

Systematic attempts at proper concentration by modern methods do not appear to have been made, in consequence of which a Newberry-Vautin chlorination plant erected at the Thames had to shut down for want of being able to obtain a regular supply of concentrates. The mill results seem to show that only that part of the gold which is readily amalgamated is caught, while the silver contents escape.

The silver in the bullion is not in fact due to the amalgamation of the silver minerals in the ore, since the proportion of silver to gold in the bullion corresponds with the native gold, which like that of Transylvania, is of very low caratage.

On the most favourable estimate but 50 per cent. of the gold is saved, leaving out of account the silver. Much of the gold and nearly all the silver is carried away with the slimes, which being carried out along the foreshore have produced an accumulation of tailings, estimated to amount to at least a million tons, carrying half an ounce of bullion per ton.

That this is not an exaggerated statement is proved by the success of the tailings-mills, which pay well.

The largest of these plants contain twelve Watson and Denny pans. The tailings are elevated and conveyed to them simply and effectively by a

small hydraulic elevator, the jet being $\frac{5}{8}$ inch in diameter, with a $2\frac{1}{2}$ inches pressure pipe, and 3 inches elevator or discharge-pipe. The water used is under a pressure of 60 lbs. per square inch. The launder from the upper end of the elevator-pipe conveys the tailings to wide strakes, where the poor slime is washed away, and the rough stones are picked out, before feeding the residual material to the pans. The cost of this treatment, including insurance, interest on plant, wear and tear, etc., amounts to 3s. 6d. per ton.

Mr. Rickard points out that while blankets, followed by pans, form a process which is quite ineffectual as regards saving the silver contents of the ore, it is also badly suited to extraction of any free-gold remaining in the pulp, after its passage over the plates.

The grinding action of the pans upon the sulphides forms slimes, which sicken the mercury, cause its direct loss, and further spoil its power of amalgamating.

The ore is both silver and gold-bearing. The former metal is chiefly associated with the sulphides, the latter is mostly in the quartz, while both minerals occur combined as tellurides, etc. Some separation is therefore needed between the silver and the gold-bearing portions of the ore. He (Mr. Rickard) suggests, therefore, that from plates the pulp should pass direct to concentrators, and thence to pans. The concentrators would separate out the silver-bearing minerals and some of the combined gold, and the pulp, freed from the sulphides, would go to the pans, which would complete the extraction of any free-gold remaining.

This combination process would obviously be an improvement on present methods, without involving much expense.

The work might be further lightened and improved upon by introducing classifiers of the Spilzlütte type, between the amalgamating-tables and the concentrating-machines. The handling of the concentrates would then be by an auxiliary process, and would not interfere with the present amalgamation upon the tables.

The use of Muntz metal in place of copper, has been referred to elsewhere, and its first introduction for the purpose took place in this district in 1875. Owing to a scarcity of copper, the local ironmongers, who imported it for sheathing ships' bottoms, sold it as a substitute. Its use rapidly spread, though the sheets locally used are too thin for the purpose, being of the thickness known as No. 18. They last, however, from three to five years.

Muntz metal as applied to amalgamation has been found to possess the

following characteristics :—It does not absorb amalgam like copper, the latter requiring to become thoroughly coated before the plates will work well.

Muntz metal has very little absorbing power over the mercury, and the amalgamation is relatively to copper, very superficial. Hence the amalgam formed on the Muntz metal is more readily detached, facilitating the clean-up. A simple rubber is always sufficient, so that the use of a steel scraper is avoided.

Test crushings are more reliable in consequence, since if the copper-plates are scraped too close before putting through a fresh lot of ore, they have not a fair chance to amalgamate, while if amalgam be left on, gold is obtained which does not belong to the stone on trial. It is not so with Muntz metal, which can be readily deprived of all its previous gain of gold-amalgam without impairing its efficiency, and is therefore specially adapted to customs mills.

On the other hand, for rich ore, Muntz metal is not to be recommended, as there is insufficient body in it, that is to say it is sooner saturated with gold-amalgam than copper. In the same way silver-plated copper, will carry more amalgam, than when the plain metal is used. This disadvantage is partly to be overcome by frequent cleaning-up. When ores contain minerals injurious to the mercury-surface, Muntz metal is preferable.

Sickening is prevented in the presence of base metals by the action no doubt of the zinc, forming one of the components of the alloy, which liberates hydrogen, and so exerts a powerful reducing effect. Muntz metal plates are easier therefore to keep in order than copper. They are hardly ever affected by sickening, and verdigris (caused by the presence of impurities in the ore and battery water) is remarkable for its absence.

Formerly at the Saxon mill, a 7 lbs. jar of potassium cyanide, costing 23s., was needed monthly for dressing the plates. After introducing Muntz metal, half a bottle of sulphuric acid (costing 3s. 4d.) sufficed for the same time. With highly acid ores (heaps of waste and old mullock-tips), copper, however, is to be preferred, as a scum is formed on the Muntz metal, while the free acid in the stone tends to keep the copper clean. In both cases it must be understood, however, that the amalgamation is interfered with.

At the Cambria, a customs mill, both varieties of plates are used: Muntz metal for the top, and copper for the bottom of the table, so as to meet the requirements of different kinds of ore. The result of experience in the Thames mills has been therefore to recommend Muntz metal for amalgamating plates where poor ore is being crushed, also in custom

milling, and where the ore is charged with minerals which injuriously affect the mercury. For material containing acid waters or very rich ore, unaccompanied by a large proportion of sulphides, copper-plates are preferable. In general, Muntz is the cheaper metal of the two, it lasts longer, facilitates a rapid and easy clean-up, and requires less attention. These points should recommend it for customs mills.

In dressing new Muntz metal plates the following are the steps to be taken :—Rub the surface of the plate with fine clean sand to get it mechanically clean, then wash it with a weak (1 to 6) solution of sulphuric acid to make it chemically clean. Then start to rub in a little mercury; rubbing in one place till it bites, that is a spot begins to amalgamate. Give a circular movement to the flannel or mop; once started, the amalgamation spreads in ever-widening circles.

There is something generally to be learnt from acquaintance with each new mill and camp, each district has a lesson to give and every mill offers some suggestion.

Practice at Clunes, Victoria.

Clunes is rendered famous as the locality where Mr. J. W. Esmond discovered the first gold found in the colony, on June 29th, 1851; though not equal in importance as a mining centre to either Ballarat or Bendigo, it has done most useful work in the development of the mining and milling practices of the Colonies.

The history of the old Port Phillip batteries of the Port Phillip and Colonial Company, has had an important effect on colonial quartz reefing; a model which has left its impress on the system in vogue at the present day. At the date when crushing first commenced in May, 1857, the treatment of gold-quartz was a problem entirely unsolved, and the Port Phillip mill laid down the basis of modern Colonial ideas on the subject. While assays proved the loss in the tailings in 1861 to amount to 6 dwts. 1 grain per ton, by numerous changes (suggested by careful experiments) this loss was decreased, till in 1870, it had been gradually reduced to 17 grains per ton. It should be remarked, however, that the average grade of the ore appears to have also altered, the yield averaging 12 dwts. 20 grains in 1861, as against 5 dwts. 17 grains in 1870.

In 1864, the plant was increased to 80 heads, and the first buddles were placed in position in 1865. The first rock-breaker was then introduced. The mine paid its highest dividends (£48,271 17s. 6d.) in 1867, on ore valued at 8 dwts. 23 grains per ton, with a loss of 2 dwts. 7 grains in the tailings. While from 1857 to 1881, it produced 1,204,908 tons of quartz, which yielded gold to the value of £1,946,989, out of which £481,455 were distributed in dividends.

The ore from the mine passes through the rock-breakers, preceded by sizing-bars (grizzleys), before entering the mill, which was built in different sections, at various periods.

No. of Heads.	Weight of Heads or Shoes. Cwts.	Date of Erection.
20	2½	1857
24	2½	1858–1859
12	2½	1860
24	3¼	1864

There are four stamps to each box. The stamp-heads and shoes are square. The mortars have a back-and-front discharge. The daily crushing capacity is at the rate of 2 tons 12 cwts. for the light stamps, and 3 tons 12 cwts. for the heavier ones. The speed is 82 drops per minute, and the height of drop is 8 inches. The issue or depth of discharge is maintained as far as possible at 4½ inches. The grating is of copper, pierced with 81 round holes per square inch.

The pyrites concentrated on the Munday buddles have amounted to ¾ per cent. of the ore crushed. Its average contents have been 4 ounces 1 dwt. 14 grains per ton. The bullion is 965 fine. The retorting percentage has averaged 38.

The business of the mill has always been carried out in a most systematic manner. The following statement of product is from the mill records for the four weeks ending May 21st, 1873:—

Where Amalgam was Produced.	Oz.	Dwts.	Retorted. Ozs.	Dwts.	Per Cent. of Total.
Mortar-box (bed)	1,466	0	673	11	59·05
Wells	708	3	249	12	21·87
Blankets	408	2	121	12	10·66
Mills	383	0	96	7	8·45
Total	2,965	5	1,141	2	

Other statistics kept were as follows:—Number of stamps, 80; tons crushed, 5,023; hours worked, 518 or 21·58 days. Average duty per stamp, 2·9 tons; yield per ton, 4 dwts. 10·12 grains; loss in tailings per ton, 20·16 grains; contents per ton, 5 dwts. 6·28 grains.

The amalgam coming from the mortar-box retorts 46 per cent.; from the wells, 35 per cent.; from the blankets, 30 per cent.; from the Chilian mill, 25 per cent. Of the total product obtained by direct amalgamation more than half came from the mortar-box, indicating the free-milling character of the ore. Of the total, 80 per cent. went no farther than the wells immediately outside the box. The above yield, deducting loss, represents an extraction of 84 per cent.

The batteries of the Port Phillip works are now idle, but the milling practice they inaugurated is seen reproduced in the newer mills of the South Clunes, and Dixon's North Clunes works.

The following table illustrates their different features. It may be mentioned that prior to 1865, the ore was calcined to render it more easily broken. This practice has not yet altogether died a well-merited death in Victoria and New South Wales:—

Name of Mill.	Port Phillip.	South Clunes United.	Dixon's North Clunes.
Number of stamps	56, 24	60	30
Weight of stamps (lbs.)	728, 896	896	896
Drops per minute	82	80	80
Height of drop (inches)	8	8	8
Depth of discharge (inches)... ...	4½	4½	7
Capacity per stamp (tons)	3	2½	3½
Total capacity (tons)...	240	150	100
Grating...		Copper-plate.	
Holes per square inch	81	100	180
Concentrates (per cent.)	¾	⅝	3
Contents of concentrates	4 ozs. 1 dwt.	3 ozs. 5 dwts.	3 ozs.
Bullion fineness (per 1,000)... ...	970	968	978
Loss of mercury per ton (grains) ...	5½	5½	5½
Wear of grating (days)	30	25	?
Water per stamp per minute (gallons)	6	8	10
Retort (per cent.)	38	42	40

At the South Clunes United mill, as the die wears down, sand is packed underneath, and when about 2 inches have worn away, a sectional false-bottom is placed underneath; in this way the depth of discharge is kept fairly constant at 4½ inches.* Each section of the false-bottom consists of a plain iron casting of sufficient length to serve for two dies, the centre die being supported by a section of half the length of the others. The rate of crushing averages 2·4 tons per 24 hours.

The screen is of copper-plate, weighing 1½ lbs. per square foot, and is perforated with 100 holes per square inch. The average wear is about 25 days, working full time. Iron punched gratings scarcely lasted a week. The concentrates have increased with the depth of the workings from ¾ per cent. to 1 per cent. The concentrates usually carry 3 ounces of gold. Lately, however, they have become poorer, 28,820 tons of ore yielding 178 tons 19 cwts. 3 qrs. of pyrites worth £560 12s. 8d. The percentage of gold in the amalgam varies from 36 to 45.

The mill-stuff is discharged into ore-bins, but the ore is fed by self-

* In California, wooden blocks, set so as to raise the screen framing, serve the same purpose.

feeders, of simple design, to the boxes, which have a front-and-back discharge.* The battery pulp passes through wells and then over blankets.

The blanket washings are treated in revolving barrels with mercury. The tailings go to Cornish buddles supplemented by ties outside the mill. Tramming the ore one-third of a mile to the mill and breaking it, costs 8d. per ton.

The feeder before-mentioned, is somewhat similar to an arrangement common in German stamp-batteries, consisting of a rod which has a round iron disc keyed to its upper end, projecting below a false or extra tappet attached to the middle stamp of a battery of five heads. The shoe (fixed to the lower end of this rod) gives a shock to the shoot leading from the ore-bin to the battery-pocket. Whenever the ore inside the box has worked down low enough for the false tappet to strike the disc of the rod, a fresh quantity of stone is thrown into the mortar.

The discs are keyed on. The order of drop of the heads is 5, 3, 4, 2, and 1. The shoes are of cast-iron, 10 inches in diameter and 10 inches high. The dies are hexagonal, of wrought-iron,† with a diameter of 10 inches and depth of 6 inches. New shoes weigh 196 lbs., new dies 140 lbs. A shoe will crush 90 tons and a die 420 tons before it is worn out Cast-iron shoes cost 12s. 6d. per cwt. and wrought-iron dies 11s. 6d. per cwt. delivered at the mill.

The gratings are vertical and covered by a splash-box which slopes forward. The front grating-frame is 5 feet by 13 inches, while the back one is 5 feet by 12 inches. The pulp discharged from the back, is led round to the front of the box, by a launder.

The whole of the battery discharge falls on to an iron plate in front of the lip of the mortar, which is $\frac{3}{16}$ inch thick, punched with $\frac{5}{16}$ inch holes, drilled at the four corners of a square inch, and is called the distributor. Its function is to spread the ore over the whole width of the tables and wells, and to catch any of the coarse stone in the battery-bottoms in case of a screen happening to break accidentally at the bottom, which is liable to occur with iron screens with a low discharge, but is generally an indication of over-feeding. In cases of this sort, where amalgamated plates are used under the lip of the box, it may save amalgam from being scraped off and lost, and as pointed out elsewhere it has a tendency to form

* The use of a double-discharge is limited to ores which cannot be amalgamated in the battery with advantage. This is a case in point, another one is with a high sulphide ore, especially if it contains brittle sulphides, which from over stamping are liable to be slimed.

† In America, steel shoes and iron dies are used in some mills, it being found that the iron die wears more evenly than a steel one would do.

thick rough bosses and ridges of amalgam, for a width of several inches, where the streams of water fall on the copper-plates.

At the South Clunes mill, the pulp from the distributor is spread over a plain wooden apron 20 inches wide and 2 inches thick which further aids its distribution. Two wells succeed this, which are guarded from theft by a wooden grating kept under padlock. The first well has a drop of 10 inches and a depth of 4 inches. It holds 50 lbs. of mercury, and the pulp in passing through it, is forced under a narrow upright board (running lengthways along the well close to its edge, on the side nearest the battery) in contact with the quicksilver. The second well, which follows immediately after, has a drop of 8 inches, is 4 inches deep, and also holds 50 lbs. of mercury. These wells, including the lip, are of cast-iron, and have a curved inside contour. They are sunk into the wood of the frame holding them. Iron wells appear to have a tendency to keep the mercury quick and lively. At Clunes they have an inside diameter of 3 inches, and are placed so as to have a slight slope to one end, where a tap-hole facilitates the removal of the mercury at cleaning-up time.

The pulp passes from the wells to the blanket tables, which have a width that takes in two batteries of five heads each. This width is sub-divided by seven partitions, 18 inches wide and 12 feet long. The grade is $\frac{3}{4}$ inch per foot. Then follow five circular improved Cornish buddles (Munday), and finally the tailings pass over ties outside the building. These last have a length of 20 feet and a fall of 1 inch per foot. The gold saving is effected by the mortar-box and wells, and indirectly by the blankets, buddles, and ties. No mercury is employed in the box, which the use of copper-gratings would in itself prevent, while the very free character of the gold, does not necessitate its use at this stage of the treatment. The mortar-box is roomy, and gives the gold an opportunity to separate from the pulp by the action of gravity alone. The interior length is 58 inches, interior width 16 inches, width between dies 1 inch, with an extra inch between the side dies and the ends of the box. Distance from dies to back of mortar 4 inches, from dies to screen 3 inches, and between centres of dies 11 inches.

In cleaning-up, the gratings are removed, and the material inside and around the dies is shovelled into buckets and passed over a strong wire riddle* 2 feet in diameter and No. 4 mesh. One of these lasts a year. The roughs from this operation go back to the mortar-box to reset the dies before restarting, $1\frac{1}{2}$ to 2 buckets being obtained at each fort-

* In Queensland, they are generally run through a tom or rocker, and the pyrites caught in the riffles is ground in pans.

nightly clean-up. The fines are sifted into a blanket-tub, and then introduced into an amalgamating-barrel. There are five such barrels in the mill, with a capacity of 54 gallons each. They make 16 revolutions per minute and are worked from 8 to 12 hours, 10 hours being the average time. The water is used cold; 75 lbs. of mercury are added to each charge with a bucket full of wood-ashes. When the amalgamation is finished, the contents of the barrel are discharged into a wooden tank, and passing through a perforated plate, flow over three drop-wells, with drops of 12 inches, 9 inches, and 6 inches respectively. Most of the amalgam is caught in the top well, the third is merely a safeguard. This disposes of the treatment of the residues.

The battery-wells are cleaned up once a week and the amalgam squeezed. The skimmings taken from the surface of the wells (largely consisting of heavy pyrites) are ground and amalgamated in three 3 feet berdans. The blankets are washed in tubs, the first row every hour, the second every alternate hour, and the third every third hour. With rich ore the washing is done more frequently. The blanketings caught, are treated in amalgamating barrels* in much the same way as the mortar-box residues. The material collected by the ties (straight troughs, in which heavy pyrites, etc., is settled by gravity), are also treated in the barrel. The tailings from all the barrels go to buddles. The concentrates thus obtained are roasted in a reverberatory furnace, then ground in a Chilian mill with the addition of mercury, by which the gold is amalgamated. The total yield of the mill in one month, crushing 2,973 tons of stone, which yielded 981 ounces 19 dwts. 12 grains of gold was thus distributed:—

	Amalgam. Ozs.	Dwts.		Bar-gold. Ozs.	Dwts.		Retort Percentage.
Mortars	955	5					
Wells	644	19	...	840	19	...	36 to 48
Blankets (by the barrels) ...	364	15					
Skimmings (by the berdans) ...	167	14	...	53	2	...	32
Tailings (by the ties and barrels)	24	10	...	7	5	...	30
Concentrates (pyrites, 17 tons)	310	4	...	80	13	...	26

Neglecting the concentrates and tailings, of the total amalgam caught, the percentage is therefore thus distributed:—Mortars, 44·8 per cent.; wells, 30·2 per cent.; blankets, 17·1 per cent.; skimmings, 7·9 per cent. The loss of mercury for the past seven years has averaged 5½ grains per ton of ore crushed. Occasionally the loss has risen to 1½ ounces per ton of ore, due to formation of copper-amalgam, which like lead-amalgam floats on

* In California, an Attwood amalgamator sometimes serves for the amalgamation of the blanketings.

the mercury, and is readily carried off with the tailings. The presence of the copper cannot be attributed to the copper gratings, but to particles of native copper in the ore. At one time, 80 ounces of copper were collected in one month from the skimmings of the wells. The total consumption of mercury (inclusive of treatment of pyrites in Chilian mills) during eleven months from July 1st, 1891, to June 30th, 1892, amounted to 3,302 lbs., when crushing 309,400 tons of ore, equivalent to a loss of 3 dwts. per ton of ore.

Up to the year 1879, according to information furnished by Mr. Hewitson, the manager, the gratings used were imported from England, being made of copper $\frac{1}{16}$ inch thick, drilled with 81 holes per square inch. When in full work the imported grating lasted twelve months, 2,200 tons of ore passing through each grating. At the Port Phillip mill, owing to its smaller area, the life of a grating reached 1½ years. The protective tariff caused the imported grating to become too expensive, and one of domestic manufacture took its place. This wore for less than half the time, but six times as long as ordinary punched iron. The experience with the present lighter type of copper grating has been very good. During the last seven years, 258 gratings have been used up. Their cost was £197 6s., and during that time 181,792 tons were crushed, or at the rate of 355 (long) tons during the life of a grating. It was found that ordinary punched Russian iron lasted only one quarter the time of the new style of grating. Baize is used for the blanket strakes; the cost of this item in one year was £49 7s. 2d. During the same period the wages at the mill amounted to £1,306 4s. 9d., treating 28,820 tons of ore, or 10½d. per ton.

The total cost of milling, including supplies, wear and tear, treatment of pyrites, etc., amounted to 2s. 3d. At Dixon's North Clunes mill the front grating is copper with 180 holes per square inch, while the back grating is brass wire with 230 holes to 240 holes per square inch. They have 6 patent Munday buddles with iron scrapers, 2 to each 10 stamps. The pyrites is washed and treated in a Chilian mill at a cost of £1 16s. 11d. per ton. The roasting of 85 tons 1 cwt. cost £89 6s. 7d., and grinding £67 15s. 11d., or a total cost of £157 2s. 6d.

The smaller capacity of South Clunes United mill, as compared with the other two referred to in the table, may be accounted for by the absence of a rock-breaker.

The finer gratings of the Dixon's North Clunes mill are offset by the less depth of discharge of the other batteries. The retort percentage of the North Clunes mill, notwithstanding that the gold in the ore is of a coarser

nature, is not quite so high as that of the South Clune's, owing no doubt to the finer size of grating. The large quantity of water used in the mills of this district is accounted for by the double-discharge of the mortars and the use of very wide blanket-tables.

The loss of mercury is exceedingly low: 5½ grains of mercury per ton being probably one of the smallest on record,* this may be attributed to the avoidance of the chief cause of loss in an ordinary mill, viz.: flowering in the battery, which cannot of course happen here.

The ore treated in these mills is broken from quartz-veins, traversing slate and sandstone-beds. When sent to the mill, the quartz is accompanied by a small admixture of country rock. The quartz is white, often honeycombed, and sometimes saccharoidal. The gold is coarse, often of very high caratage, frequently visible to the naked eye, and arranged for the most part along the faces of small fractures, or seams in the quartz; a blow therefore tends readily to detach it. Occasionally the percentage of mullock (waste rock) increases considerably, and the gold is accompanied by pyrites, chiefly composed of arsenic and iron sulphides, or occurs in a matrix consisting of slate and quartz intermixed.

The Port Phillip and Colonial mill-book shows that the proportion of the total yield of gold coming from the mortars and wells steadily declined from 1868 (when it stood at 87·03 per cent.) to 1879 (when it stood at 68·49 per cent.). On the other hand, however, the yield from the blankets and concentrates increased correspondingly. The explanation lies in the fact that as the mine workings reached a greater depth, the ore, by the steady increase of the pyrites it contained, became less free-milling.

This reasoning is confirmed by the returns, for while in 1866, the yield of concentrates amounted to 268 tons, averaging 2 ounces 19 dwts. 4 grains from the crushing of 59,578 tons; in 1879, the pyrites amounted to 421 tons, averaging 4 ounces 15 dwts. 20 grains, resulting from the treatment of 56,766 tons. In 1870 the use of blankets was discontinued, but in 1873 it was resumed; in the interval the yield from the Chilian mill increasd considerably. In 1865, the ore yielded 7 dwts. 13¾ grains per ton; of which amount the boxes yielded 63·60 per cent.; the wells, 22·09 per cent.; blankets, 10·55 per cent.; mills, 3·76 per cent.; and mills and blankets, 14·31 per cent. In 1872, the ore yielded 4 dwts. 17¾ grains per ton; of which amount the boxes yielded 64·48 per cent.; the wells, 21·60 per cent.; the blankets, 1·06 per cent.; the mills, 12·86

* The record of greatest waste, it is stated, occurred at a mill in the Thames district, where 1 ton of mercury was used up in two weeks by a mill of 20 heads.

per cent.; and the mills and blankets, 13·92 per cent. In 1879, the ore yielded 8 dwts. 19¾ grains per ton; of which amount the boxes yielded 57·99 per cent.; the wells, 10·50 per cent.; blankets, 12·84 per cent.; mills, 18·67 per cent.; and mills and blankets, 31·51 per cent.

To consider the method in practice, it will be admitted that the use of costly chemicals in milling is as far as possible to be avoided. Mercury is generally a large item, and since 55 to 65 per cent. of the gold can be arrested in the box, without its use; the practice of the district in dispensing with it in the box, appears correct and advantageous. Under the stamps, it is liable to be flowered, *i.e.*, broken up into minute globules, which, collecting impurities, become covered with a film which makes them refuse to coalesce, causing them to be carried away on the surface of the water. With this loss of mercury there must also be a loss of gold, particles of which have entered into amalgamation with the escaping globules.

The absence of amalgamating-plates is remarkable, but in view of the character of the ore, Mr. Rickard considers it correct. Wells, as he states, are excellent-gold savers for ore of this type, in which the metal is both free and coarse. He points out as their advantages that they require less attention, their first cost is less than that of plates, and they are less affected by the occasional presence of minerals in the ore, injurious to amalgamation. Blankets, when intelligently used, are among the best of the simple contrivances known to mill-men, given as they are at Clunes plenty of width. At South Clunes there is a clear blanket space of 10½ feet. Ordinarily the slope of the blanket-tables would be 1¼ inches to 1½ inches per foot; but at Clunes, owing to the use of plenty of water due to the double-discharge, they have a pitch of only ¾ inch per foot. This is in itself an important factor, apt to be overlooked.

The after-treatment in the barrels may seem crude, but practice has shown it to be effective. The bad custom of putting pieces of iron into the barrel, with the idea of mixing and grinding-up the pulp, is here avoided, saving a large loss in flowered mercury. The means taken to keep the discharging depth of the mortar constant, is a factor of importance in the method pursued, which does credit to the practice of the district. The self-feeders do their work well, and though not perfect, are a great improvement on the bad and irregular hand-feeding which prevails in the majority of Colonial mills. The concentrating machinery may with reason be considered somewhat out of date, but the modified Cornish buddles in use, are doing excellent work. The use of the double-discharge mortar increases the crushing capacity of the mills, but also requires a much increased supply of water.

Speaking generally, the treatment the ore undergoes is remarkable, most of all on account of its simplicity, but so is the ore; and in this way the local practice carries out the first postulate of intelligent milling, viz., that the treatment should be varied according to the character of the ore to be dealt with. After a careful examination of the ore mined at Clunes, and of the milling it is subjected to, it is possible only to speak in words of commendation. To a mill-man it is almost solitary among the gold-mining districts of the Colonies, in being a quartz-milling centre which does not leave a feeling of dissatisfaction, and an impression of dissapointment on the visitor. You may visit mills in the most distant parts of Australia, and almost without exception, whenever you find good intelligent milling (and that does not happen often enough to be monotonous), the knowledge and experience of the individual in charge, have been obtained at Clunes.

The Port Phillip mill was the first to introduce the system of taking daily assays as a check on the work done in the mill.* In this respect the Clunes mill is still unfortunately a striking exception. In another department this mill was also a solitary pioneer, the first rock-breaker being introduced by the Port Phillip in 1865. Yet in Victoria to-day, there are only twelve rock-breakers!

Mr. Rickard expresses the belief that the work done by the Port Phillip and Colonial Company's mill, has been of more wide-reaching usefulness and more permanent benefit to the mining industry of Australia and New Zealand, than that of any other company which has gone into operation since the days of the discovery of gold, and records his conviction of the debt which quartz-milling in the colonies owes to its manager, Mr. R. H. Bland, who started its operations in 1856, conducted the numerous and valuable experiments which did so much to establish the correct basis of milling practice, and to-day still assists the industry by his sterling good sense.

* Mr. Rickard mentions as another instance, the mill of the Harrietville Gold Mining Co., Ld.; and it was also done at the Disraeli mill in Queensland, when under the writer's charge.

(*To be continued.*)

THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.*

By A. G. CHARLETON.

Part VI.—Gold-Milling.—*Continued.*

Practice in Dakota.†

The gold of the district is found in quartz and pyrites finely distributed through vast masses of mica and amphibole-schists, argillites, and phyllites, and also impregnating the schists themselves. The gold-belt embraces the sections of Lead City, Terraville, and Central City. The principal associated mineral is iron pyrites, with some arsenical pyrites, garnet, and asbestos.

The ores from open cuts and the upper levels are more free-milling than those from the mine workings below the water-line. Hence the mills running on oxidized ore have tailings valued as low as 1s. 0½d., while tailings from unaltered ore run up to 9s. 4½d. per ton. By watching the pulp when it flows down the plates, it can readily be determined whether it comes from the higher or lower levels. In the former case it will generally have a brownish-red colour, and in the latter it is of a bluish-grey. The amount of free gold in the ore varies with depth, and probably 16s. 8d. per ton is a near average of its value.

To determine the amount of free gold in the ore the following method is practised :—Samples are taken daily from the various workings in the mine and sent to the sampler, who crushes and pans them and estimates the gold in each pan. Every valuation thus made is booked, and at the end of the month the average is taken, and compared with the output of the mill, and the amount of gold recovered thus approximately determined. The mode of operation practised by the sampler is very simple. The sample, weighing say 10 pounds, is emptied into a 4 gallons bell-shaped mortar (13½ by 12½ inches). From it 2 pounds are then

* *Trans. Fed. Inst.*, vol. iv., pages 233 and 351 ; vol. v., page 271 ; vol. vi., pages 69 and 295.

† The writer is largely indebted in this section to a paper by Mr. H. O. Hofman, on "Gold-Milling in the Black Hills."—*Trans. American Inst. Min. Eng.*, vol. xvii., page 498.

transferred into a second mortar of the same size with a wooden lid, and pulverized wet to a fine pulp by means of a small steam stamp, which is in reality an old power-drill fitted up for the purpose. When sufficiently fine (as judged by the ring of the pounding-stamp) the pulp is panned until all the pyrites and heavy sands are washed off with the tailings, and only the free-gold remains behind. The sampler of the Homestake Company pans from 50 to 55 samples per day. Great skill is acquired in thus estimating the value of the ore, the sampler being able to make eight or ten valuations per hour. As these are the only determinations made, the amount of non-free-milling gold which enters the mill is not known.

The percentage of sulphides from several determinations that have been made varies from 2½ and 3 to 6 and even 10 per cent. The assay value of pure concentrates freed from rusty gold, or gold included still in the quartz, has been shown to vary from 16s. 8d. to £16 17s. 6d., the average being about £5 4s. 2d. per ton.

As the ore is finely disseminated throughout the entire vein-matter, comparatively little sorting can be done in the mine. There occur, however, in many parts of the veins igneous intrusions, locally called porphyry, which form barren horses. When the Nevada system of timbering in square-sets was exclusively in use no distinction was made between mill-rock and waste, but it was considered cheapest to run it all through the mill. Latterly, however, it has become the custom to fill the chambers formed by this timbering with waste, and to hoist the excess to the dump; but large quantities of it are still crushed.

The auriferous lodes and gravels were discovered in this district in 1876, when a rush to the Black Hills took place.

The seven principal mills are the Homestake, Golden Star, Highland, Deadwood, Golden Terra, Father de Smet, and Caledonia. The first six are owned by five separate companies, but are all under the management of the Homestake mill superintendent, and all the working details are much on the same model. The Caledonia mill works on different rock, and differs from the others in details of plant.

The crushing is all done by rock-breakers and stamps. The ore arriving at the ore-floor at the highest level of the mill is discharged from side or bottom-dump cars over grizzlies, which divide it into lumps (going to the breaker) and fines (falling into the bins). The lumps are crushed in the rock-breakers and join the fines, and the mixed product, passing through shoots, goes to the automatic feeders, which deliver it to the stamps.

The Caledonia mill has blankets on the lower end of the apron-plates, below the copper plates, to catch any coarse heavy particles. In the other mills the pulp passes direct from the apron-plates to the mercury-traps, and through them on to sluice-plates. From the traps placed at the end of these the pulp runs into one main sluice, which may have again one or more mercury-traps before the pulp is allowed to finally run to waste.

Amalgamation begins in the mortar (which is provided with inside copper plates, mercury being added to the box at intervals), and is continued outside on the apron-plates; any mercury or amalgam escaping these appliances being caught in the mercury-traps* lower down, which are used to supplement them.

The aim in Dakota is to crush rapidly to the desired degree of fineness and arrange the amalgamation so that it shall be adapted to the large amount of pulp produced. The distribution of power in the mills is of three different types, represented respectively by the Homestake, Golden Star, and Highland mills.

In the Homestake mill the continuation of the engine-shaft forms the one-line shaft of the mill, and is placed on the battery-sills. This is a cheap construction, and gives a solid foundation for the boxes in which the line-shaft rests. The shaft is kept in line by the even pull of the long belts on each side of it, running at an angle of about 30 degs. between the bins in the centre of the mill. The disadvantage of having to stop the mill if anything happens to the line-shaft may be dismissed as insignificant. For minor repairs, however, it is in an obscure place, running behind the batteries.

In the Golden Star mill the power is transferred from a small main shaft to two line-shafts on the cam-floor, which is nearly on the same level. This arrangement is usual in Pacific coast batteries.

In the Highland mill the small main shaft is placed between the cam-floor and the battery-floor, and is connected with two line-shafts† placed on the battery-sills behind the mortars, which are set back to back with the bins between them. Two lines of shaft are used, in consequence of the power required for a 120 stamp. The disadvantage is that the pull of the belt on one side only of the shaft, has a tendency to drag it out of line and cause excessive wear and tear.

* An illustration of this apparatus will be found in the *Eighth Annual Report of the California State Mineralogist*, page 711.

† These are coupled up in sections, 7 inches in diameter near the engine, reduced at the opposite end to 4 inches.

As to placing the line-shafts in front of the batteries on the cam-floor (as in the Golden Star mill), or behind them on the battery-floor (as in the Highland), there is a diversity of opinion. The former arrangement gives the best light, and makes the shafting easily accessible. The power from the engine-shaft is transmitted to the line-shafts, and from these to the cam-shafts by horizontal belts which require no tighteners and last longer. On the other hand, the boxes of the line-shafts rest on traverse sills on the cam-floor which, although braced and otherwise strengthened, do not afford as solid a foundation as when fixed on the battery-sills. On the whole, however, this disadvantage appears more than counterbalanced by their greater accessibility and the smaller wear and tear of belting.

On the battery-sills behind the mortar, the line-shaft is in darkness and exposed to the trickling of water and the abrasion and dirt of fine ore falling from the feeder-floor, whilst the belts connecting the main and line-shafts and the latter with the cam-shafts, are short and highly inclined, requiring powerful tighteners.

The relation of the horse-power of the engine to each stamp averages about 1·7 to 1 for the seven mills. This low figure is due to the large number of stamps in each mill (80 to 120) which is much above the ordinary number.

The Father de Smet mill is built with the batteries on opposite sides of the building face to face, an arrangement which is said to have the disadvantage of rendering it rather dark.

The Golden Star and Highland mills have each 120 stamps, and the Father de Smet mill 100 stamps of 850 lbs., dropping 85 times per minute, with a fall of 9 inches, arranged in batteries of five. The Homestake, Deadwood, and Golden Terra mills have each 80 stamps, with the same weight of stamps and height and number of drops. All these mills crush on the average about the same weight of ore per stamp head, viz., 4·5 tons per 24 hours. The Caledonia is a 60 stamp mill with 850 lbs. stamps, dropping 12 inches, 74 drops per minute, and it only puts through 3·3 tons of ore per 24 hours.

The water supply is furnished by ditch companies at 50 to 57 cents per stamp per diem. A regular supply is an essential for milling. When it becomes scarce in winter it is supplemented by pumping from the Homestake and Deadwood Terra shafts.

The Highland mill would be obliged to stop four months each winter if the tailings of the Homestake, Golden Star, and Highland mills were not settled and the clear water pumped back into the mill supply-tanks. The

method by which this is done is of interest. The tailings of the three mills named are discharged into Gold Run creek. A little way down, where the creek broadens it is closed by two dams, one below the other, forming an upper and lower reservoir. The former overflows into the latter, which is four times its size, and this in turn overflows into the bed of the creek. The dams consist of cribbing filled with waste rock, lined on their upper sides with planking to make them watertight. Down the middle of the face of the upper dam runs a wooden box or launder, three sides solidly planked, with the fourth open; when the reservoir is to be filled the box is closed by heavy transverse strips of planking. The object of a number of pieces is to discharge the water gradually, which is done by removing the pieces one after the other as the water is lowered, so that the sands may be kept in suspension and carried through the culvert. Were the box open from the bottom or to its full height at once, the sands would be carried into the culvert in such quantity as to close it. This culvert in which the box ends, passes through the dam and under the large reservoir and lower dam to the bed of the creek below. The lower dam is arranged in the same way. When the reservoirs are not in use the water of the creek passes off through the culvert. When they are to be filled the boxes are closed, and the water accumulates in the upper reservoir until, after 6 hours, it overflows, leaving all the coarse sands in the upper reservoir, and carrying only the finer slimes, which settle in the lower one. From this the clarified water is pumped at the rate of 60 cubic feet per minute into the Highland tank 200 feet higher.

The coarser sands are removed from the upper reservoir every 24 hours. In order to do this the traverse planks closing the discharge are removed one after another, and the water passes off, carrying the sands with it. As this process takes 4 hours, and the filling 6, there are 14 hours of overflow into the lower basin, where the slimes settle. These are removed once in two months in the same way.

In Queensland, Australia, where water has to be rigorously economized, a reservoir is frequently built in a suitable position at the side of a creek, so as to enclose a large area. Openings are then made in the artificial embankment which takes the place of one bank of the creek, and lock-gates are thrown across the stream forming an extension of the embankment at the lower end of the reservoir. In the rainy season these gates are closed temporarily so as to raise the water to the level of the openings in the embankment, through which it finds its way into the reservoir until the latter is filled. The water accumulated in this way is used as a

reserve in the dry seasons, at which time, if the sides of the creek are abrupt enough to admit of doing so, large dams of sand formed of tailings may be thrown across the bed of the stream to back up any water that can be accumulated behind them. The accumulated tailings are washed away in the rainy season when the flood-gates are opened. The water, notwithstanding that it carries considerable silt, has often to be used over and over again by pumping it from the reservoirs with a tailings-pump, which, in order to prevent the plunger from being cut to pieces, has a small stream of water introduced under pressure by a small pipe into the plunger-case.

In some Queensland mills, the system adopted is to construct a breast-dam running right across the creek, with a gate in the middle through which the silt which accumulates, partly in times of flood, partly from settling the tailings, can be flushed out. At the end of the rainy season, it is filled with water, and the mill draws its supply from this reservoir by means of a pump. The mill-tailings are discharged immediately below the reservoir through box-launders put together in short sections, the lower end of one entering the top of the section below. After a heap of tailings has accumulated against the back of the dam, at the side of the creek over an area of 5 or 6 square yards, an accumulation which can be facilitated by changing the position of discharge of the launders, the flow is diverted to one side until the heap has drained and settled enough to allow of a shallow pit about 1 foot deep being dug in the surface of the heap (which is roughly levelled), leaving a wall of sand (tailings) round the edge. The mill tailings are then allowed to run into this temporary basin, and the sands settle in it, whilst the partially clarified water is returned by a short launder through a hole in the face of the dam (near the top) into the reservoir. Whilst the first basin is being filled up, the tailings-man is engaged in digging out a second one at the side of the first, into which the tailings are diverted when the first one becomes partly filled with sand. The heap forming the foundation of the basins is constantly extended outwards by throwing with a shovel the sand from the basin to the outside of the wall of sand, which forms a natural talus sloping outwards. It is of the greatest importance when this method is used to keep the level of the basins as high as possible, with a slight rise outwards as they are extended into the creek, otherwise the water cannot circulate backwards at a sufficiently high level to enter the reservoir at the point where the opening is made for this purpose.

The fuel for the mills of the Homestake management is supplied by the Black Hills and Forte Pierre Railroad Company. This road, with

about 30 miles of 3 feet gauge track, runs along the divide between Gold Run and City Creek, terminating at a point about 14 miles south of Lead City. The whole section was originally heavily wooded, but has been quite denuded by the constant demands made upon it. The railroad is very winding, and is quite a feat of engineering. It runs down the slope into Whitewood Creek and up the opposite height, till it finally reaches the point where timber is still to be obtained. This road, as soon as it is open in the spring, is employed in transporting the timber which has been cut and stored along its line, and is in constant use till it becomes blocked with snow, generally from January till April. It has three branches towards the three towns where the mills are situated, and communicates directly with these by long wooden shoots, down which the timber is discharged. These shoots are 700 to 1,500 feet long, running down the slope of the mountain, they are 25 inches broad by 12 inches deep, and are made of 4 inches planking. The bottom and 9 inches of the sides are lined with ¼ inch iron plates. The fall of the shoot is 6 inches to the foot until the curve begins, when it is 4½ inches. This continues to the nozzle, which is elliptical. When the shoot is in use a small current of water is passed through it, to prevent the iron from becoming too hot and to act as a lubricant. The cord wood unloaded at the top of the shoot passes down the incline with great velocity. At the nozzle it is deflected from its course, and through the momentum obtained in the downward passage it shoots up into the air and drops some distance off on to the wood-pile. In order to discharge the wood on a large area and to stack it, the nozzle is made movable.

The price of wood in the district is £1 5s. per cord. The water is warmed, as elsewhere mentioned, to keep it from freezing in winter, and fireplugs with hose-attachments are placed at intervals in the mills as a safeguard against fire.

To reduce the cost of repairs, which is a heavy item, the Homestake Mill Company has a foundry where rock-breaker shoes and dies, pitmen and toggle-plates, mortars and dies, boss-heads, tappets, thimbles for fingers, cams and hubs of cam-shaft pulleys and shaft-boxes, etc., are cast from Nos. 1 and 3 foundry-iron and worn-out castings. The castings are made in sand, with the exception of the rock-breaker shoes and dies and faces of battery-dies, which are chilled. All the necessary repairs are executed in the machine-shop, which is a very complete establishment. The six Homestake mills use the same patterns for all parts requiring frequent renewal, which reduces the amount of material kept on hand, and labour and cost of repairs.

The Grizzlies relieve the rock-breakers of the ore which does not need crushing. They are 3 to 4½ feet wide, 10 to 14 feet long, and set at an angle of about 40 degs., or a rise of ⅞ inch per foot. They are made generally of wrought-iron* bars 1 inch wide and 2 to 4 inches deep, held 1½ to 2 inches apart by three or four, sometimes five, 1 inch iron rods, provided with thimbles at proper intervals to keep them apart. The Father de Smet grizzlies, which are 4½ by 12 feet, with 24 bars 1 inch by 2 inches set 1½ inches apart, weigh 2,040 lbs. The grates last about four years.

Rock-breakers.—No. 5 Blake rock-breakers are mostly in use, with jaw (opening 9 inches by 15 inches) set to crush from 1½ to 1¾ inches, run for 20 hours out of the 24 ; this size is calculated to serve 20 stamps. If one-fourth of the ore passes through the grizzlies and 20 stamps crush 90 tons in 24 hours, the amount crushed by one such Blake breaker in 20 hours is 67½ tons or 3·4 tons per hour. This small figure as compared with the nominal capacity of the crusher, *i.e.*, 7 tons, is due to the delay in breaking up the ore so as to enter the jaws. The advantages of the Gates crusher in this respect have been previously mentioned. The No. 6 Gates crusher in use at the Caledonian mill is said by Mr. T. L. Skinner, the superintendent, to have saved him £5 12s. 6d. per day.

Mr. Hofman advocates for a large mill a still larger Gates crusher (No. 8 with receiving openings 18 inches by 48 inches), set to crush coarse, discharging into two No. 6 crushers set to crush fine. Thus the largest pieces of rock that any man could handle would pass direct into the crusher without hand-breaking.

Ore-bins.—The ore-bins are triangular in section, with one vertical side facing the battery and reaching down to the cam-floor. Just above the latter are the openings (one for each feeder) through which the ore passes down into the shoots terminating in the hoppers of the feeders. The discharge can be regulated by a sliding-gate, with a rack worked by a pinion keyed to the shaft of a small hand-wheel. In a double mill the inclined bottoms of the two bins diverge, leaving an open space between them having the shape of an inverted V. The bottoms of the bins, 3 inches thick, are made of 1 inch board, running lengthways, with 2 inches planking set crossways on them, but this is not the best arrangement. The bottom and sides are strongly braced with timber. The upper part of the bottom on which the ore drops from the grizzlies and crushers is

* Hard cast-iron is being introduced for this purpose in some mills.

lined with iron to prevent it from wearing out faster than the lower parts, which last five to six years. There are no compartments or divisions to direct the ore to the discharge-openings, as they are unnecessary. Ore-bins should be given as large a capacity as possible, so that, in case of accident at the mine or to the rock-breaker the mill need not be stopped. The Highland mill has the largest bin capacity of the group under review, except the Father de Smet mill. In the Highland mill the horizontal distance between two sets of batteries is 46 feet, and the vertical height is 22¾ feet. In the Father de Smet the apron-plates are overshadowed by the bottom of the bins, which neutralizes the advantage of facility of supervision claimed for this arrangement, by rendering the building dark.

Feeders.—The feeder may be arranged so that the lip of the feeder reaches into the feed-slit of the mortar, but at the Caledonia mill they discharge with a small inclined iron-lined apron which leads to the mortar. By this arrangement a little more room is left between the feeder and mortar, and the feed-opening can be longer and narrower, distributing the ore more uniformly under the stamps. The Homestake mill mortar feed-opening is 24 inches long and 4½ inches wide, while at the Caledonia it is 52 inches long and only 3 inches broad, occupying the whole length of the mortar. The Hendy Challenge and Tullock automatic feeders are in use, the Challenge feeder being the most desirable for wet ores.

Both right and left-hand feeders are used, the bumper-rod standing between stamps 1 and 2 or 4 and 5. Some of newer pattern are made with the bumper-rod next the central stamp (central feeders). The rod is guided from the cam-floor, passing through a hole in a board fixed to it. The sheet-iron plate below the Tullock feed wears out quickly (with Homestake ore in two years), but is cheap and can be patched or renewed by a blacksmith. The circular cast-iron carrier-table of the Challenge feeder lasts seven years with the same class of ore, but is costly; and if anything goes wrong with the gearing, it requires a fitting shop and machinist for its repair.

Foundations, etc.—A good foundation is the starting-point of every piece of good engineering work, and nowhere is it more necessary than in a stamp-battery. A rectangular pit, 11 to 14 feet deep, is excavated to receive the mortar-blocks, made sufficiently long and wide (4 feet by 6½ feet) to leave a space of about 24 inches round the block. The bottom is then carefully levelled and some sand tamped down, or concrete is run

in. On this are placed two layers of 2 inches planks, spiked cross-wise to each other, and then the planks which form the mortar-block. The latter used to be placed in the pit, and the uneven tops were afterwards sawn off level. Now care is taken that this 4 inches wooden floor shall be accurately horizontal, and that the distance between it and the bottom of the mortar shall correspond with the length of the mortar-blocks. The top of the block is planed. By employing this flooring the time required to replace a mortar-block is reduced from six or seven days to five.

The mortar-blocks used in Dakota consist of planks from 11 to 14 feet long, depending on the depth of pit, of varying width, and not more than 2 or 3 inches thick, as it is difficult to find wood of greater thickness and yet sound throughout. They are spiked together, and fastened above and below with binders bolted to each other by transverse rods, the upper binders (8 by 12 inches) being even with the top of the mortar-block, and the lower binders (12 by 12 inches) 3 feet lower down. The space round the block is then carefully filled with rock and tailings up to the level of the mud-sills, which are about 4 feet below the bottom of the mortar. When the top of the block has been planed off level, a sheet of rubber-cloth $\frac{1}{4}$ inch thick is placed over it and the box put in place. Through the four holes in the flanges on each side pass 8 bolts from 3 to $4\frac{1}{2}$ feet long, and $1\frac{3}{8}$ to $1\frac{1}{2}$ inches in diameter, by which the mortar is held down.

In placing the planks forming the block and adjusting the holding-down bolts an improvement has been introduced at the Homestake mills. The planks which always stand on end were formerly so placed that their width was parallel to the short side of the mortar. The holes for the 8 bolts were then bored into the block from above, and at a suitable distance below recesses were chipped out to receive the nuts which held the lower ends of the bolts. Now the planks are so placed that their width is parallel to the long sides of the mortar. The bolts are only threaded at their upper ends, and end in an eye at the bottom, 2 inches bolts pass horizontally through these loops from side to side of the block, the planks on the two sides of the block where the bolts pass down, being cut out to receive them.

In addition to the mortar being more securely and evenly tied to the block, this makes the replacement of a mortar-block if required much easier. The pit need only be dug out in front of the mortar, and when the front binders have been removed it is easy to tear out the planks one after another with pick and adze. In putting in the new block, the two out-

side rows of planks, with places cut to receive the bolts, are kept ready so that only four horizontal 2½ inches holes for the rods need be bored when the planks have been spiked together.

Battery-posts are usually made of 12 by 24 inches timber with recesses cut for the boxes of the cam-shaft. They are set on the short sides of the mortar and are independent of the mortar-block, standing on the cross-sills which rest at right angles across the mud-sills; they are tied together by the upper and lower guide-timbers and at the foot by two beams bolted to them, running parallel with the long side of the mortar-block, and let into the cross-sills.

The frames are braced either from back or front of the battery, depending on the way in which power is transmitted to the cam-shaft. When the line-shafting is at the back they are braced by inclined struts at the back between the posts and cross-sills. When the cam-shafts, however, receive their motion from line-shafting in the cam-floor in front, the posts are braced by horizontal cross-beams, resting on a line of posts in front, and projecting beyond them, which are strutted in the angles, and tied together by a longitudinal cap-piece.

Guides.—The stems work in two sets of guides fixed to the guide-timbers which tie the battery-frames. The upper guides are above the tappets, the lower ones below the cam-shaft. At the Homestake mill the centre of the lower guides is 17½ inches above the top of the mortar; the cam-shaft is 4 feet higher from centre to centre, and the centre of the top guides is 3 feet 10½ inches above this. At the Caledonia mill the distance from the top of the mortar to the centre of the lower guides is 16½ inches, from these to the centre of the cam-shaft is 3 feet 4½ inches and between the cam-shaft and the top of the guides, from centre to centre is 4 feet 2½ inches. Each set of guides is provided with liners (guide-blocks) consisting of two strips of 4 inches pinewood, 16 inches deep, provided with semicircular grooves for the stems. When new, small wooden strips are inserted between the back and front of the blocks to hold them slightly apart, these strips are removed as the grooves become worn, bringing the faces of the guide-blocks nearer together. When still further worn the faces are planed to diminish the depth of the grooves, so that the stems may not be too loosely held. Each set of guide-blocks is secured to the guide-timbers with eight ⅞ inch bolts. The grooves and the guide-blocks are lubricated with a preparation of black lead and linseed oil, mixed warm so as to form a soft paste. Oak guide-blocks last 18 months, pinewood only four.

Mortars.—Two kinds of single-discharge mortars are used in the district, each being a solid casting. The sides and bottom are so thick that a lining is dispensed with, but as the feed-opening is approached the thickness rapidly diminishes. The discharge-side projects somewhat, and the other sides are vertical. The top is closed by two pieces of 2 inches planking which rest on lugs ¾ inch wide, cast on the inside of the box 2 inches below the top. These planks have, as usual, five semicircular recesses cut in them, which, when placed together, form holes for the stamp-stems to pass through. Two smaller holes are also bored in them, for two 1 inch water-supply pipes placed between stamps 1 and 2, and 4 and 5.

The mortars are set close together in pairs, 10 stamps being operated by one cam-shaft, and a passage-way is left between every two pairs of batteries.

The water-supply is furnished by a 3 inches main-pipe in front of the batteries, from which a 2 inches stand-pipe passes upward between each pair of batteries. A 2 inches horizontal pipe is connected with the stand-pipe from which four 1 inch pipes branch off at right angles, two for each mortar. A 1 inch pipe at each passage-way close to the mortar, connected with the 3 inches main-pipe, serves for the attachment of a hose for cleaning the apron-plates and other purposes.

The points of difference between the Homestake and Caledonia mortars lie in the dimensions of the lower part of the box, and the number of the inside amalgamating-plates. The Homestake mill mortar weighs 5,400 lbs., is 54½ inches high, and 54¾ inches long. The feed-opening (6½ inches below the top) is 24 inches long, 4½ inches wide, and 7 inches deep, and continues the same length inside; the incline over which the ore falls being extended to form a lip 4¾ inches wide and 1¼ inches thick, projects into the mortar so as to discharge the ore against the upper half of the stamp-head. The lower edge of this lip is 14 inches above the bottom of the mortar. As it wears out quickly, it is cast thicker in the Caledonia mill mortar.

The discharge-opening in front of the Homestake mortar is 15½ inches from the bottom, 48½ inches long, and 21¾ inches high. The frame-seat is inclined outwards about 10 degrees from the vertical, and there are grooves at its ends to receive the chuck-block, screen-frame, and curtain, which are held in place of keys. The chuck-block is secured also at the bottom by two horizontal keys supported by lugs cast on the lip of the mortar. The bottom flanges are 3 inches high, and 5 inches broad; the bottom is 7½ inches deep, and the sides at the foot of the dies are 3½

inches thick. The width inside at the bottom is 10½ inches,* its length is 50 inches, and height to issue of mortar, *i.e.*, the bottom of the discharge-opening, 8¾ inches. The inside width of the mortar at this point is 13½ inches, and at the top of the discharge-opening it is 20 inches. At the top of the mortar it is 16 inches, and the total inside height is 47 inches. The casting is ¾ inch thick from the top down to the feed-opening on the sides and front, but the back is a little thicker. The life of a mortar is about four years.

The Caledonia mortar weighs 5,700 lbs., is 57½ inches high, and 54 inches long. The feed-opening begins 15½ inches from the top, is 3 inches wide, 11 inches deep, and extends the whole length of the mortar, with a strengthening rib in the middle. Where it enters the mortar it is 50½ inches long, and 7½ inches deep, with a lip 2½ inches thick, and 8 inches wide, measured on the incline. This discharges the ore towards the head of the stamp, and protects the amalgamated plate below. The front discharge-opening (50 inches by 17 inches) is 20 inches above the bottom of the flange, and is inclined forward about 10 degs. The grooves on the sides receiving only the screen-frame and curtain are of simpler construction than the Homestake mortar. The lugs for the horizontal keys are the same. The flange round the bottom is 3 inches thick and 4½ inches wide. The mortar-bed is 7 inches thick and the sides at the foot of the dies are 4½ inches thick. The width inside at the bottom is 10 inches, the length 50½ inches, the height 14 inches to issue of mortar and pulp, where the width is 16 inches and increases to 19 inches at the top of the discharge. The top of the mortar is 13½ inches wide, the total inside height is 50½ inches, and the casting from the top to the feed-opening is ¾ inch thick. A mortar lasts six years, and wears out more at the ends than at the back. The feed-opening, for reasons previously recorded, is longer in the Caledonia than in the Homestake mortar, and its inside lip is thicker and wider in the former type, a difference which is necessitated by the use of the amalgamated plate below. For the same reason the Caledonia mortar is also made wider at the issue. The depth of the discharge-opening of the two types moreover differs. In the Caledonia mortar it is 14 inches, which represents the point of issue of the pulp, whilst in the Homestake mortar it is only 8¾ inches, as the issue is raised 16¼ inches above the dies by the insertion of a chuck-block, thus giving the shallower mortar the deeper issue.

Dies.—The dies are cast by the Homestake Company on the spot, using an iron between grey and mottled, the top of the cylindrical column

* Formerly it was 14 inches, but this reduced the crushing capacity.

being chilled. The footplate has bevelled corners, and is 10 inches long, 10½ inches wide, and 1¼ inches thick. The column or boss is 9 inches in diameter, and 5 inches high. The level of the die is 10 inches below the discharge, which is over the chuck-block. The die weighs 121 lbs. (one-seventh the weight of the stamp), and lasts about six weeks, crushing 189 tons. By that time the boss has worn down to 2 inches from the footplate, and is slightly convex, its weight being reduced to 30 lbs., it shows a consumption of 48 lbs. of iron per 100 tons stamped.

The Caledonia mill purchases its dies, which are made of chilled white iron. The footplate is 10 inches wide by 9½ inches long, and 1½ inches thick. The boss is 8 inches in diameter and 5½ inches high.

The dies in the Homestake mortar fill the bottom completely, those of the Caledonia only fit perfectly crossways, half an inch of space being left between them. From the bottom of the screen to the top of the die is 6 inches. The die weighs 160 lbs. (about one-fifth the weight of the stamp), and lasts three months, crushing 300 tons of hard rock. The boss is then worn to within 1 inch of the footplate. The worn-out die weighs 38 lbs., making the consumption of iron 40 lbs. for every 100 tons of rock.

Amalgamated copper-plates are placed along the entire length of the mortar. In the Homestake there is one plate only set in the discharge-opening. In the Caledonia a second one is used below the lip of the feed-opening.

In the Homestake mortar the chuck-block, consisting of a 2 inches planking bolted to the back of a 1¾ inches board, and extending from 2 to 2½ inches above it, fills the bottom of the discharge-opening, the ends of the plank being held in the end-grooves outside. The inside upper edge of the block is rounded off, and over this and along the inside face a $\frac{3}{16}$ inch copper plate is fastened with iron screws. The recess in front of the chuck-block on the top of the front board (1¾ inches wide and 2 to 2½ inches deep) is taken up by the lower side of the screen-frame, between which and the front board a strip of blanket is laid to form a tight joint. The recess under the chuck-block back of the front board is filled by the side of the opening of the mortar.

The screen-frame is held in place by a vertical plate of iron bolted to the centre of the front board, with a horizontal wedge driven between them. The front board is faced, in the centre of the lower half and ends, with iron-plate to protect the wood against the two vertical and horizontal wedges which hold it to the mortar.

A strip of rubber cloth is tacked to the bottom of the chuck-block, in order to make a tight joint inside between it and the flange of the mortar-

opening. Two chuck-blocks of different heights are used, one 7 inches in height used when the dies are new, and one 5 inches high inserted when they are worn down 2 inches. The height of discharge is thus kept nearly uniform. Wooden chuck-blocks last six months.

Owing to the distance between the edge of the dies and the face of the chuck-block being rather small, viz., 2 inches, it was found in the Homestake mill that the sands driven violently by the water against the copper plate scoured off some of the amalgam. Mr. Graham, the millwright, has therefore replaced the 2 inches planking to which the copper plate is screwed, by a $\frac{1}{4}$ inch iron plate, to which the $\frac{3}{16}$ inch copper plate is riveted with copper rivets. The face of the $1\frac{3}{4}$ inches front board being covered with $\frac{1}{8}$ inch iron plate, the distance between the dies and the Graham modified chuck-block is $3\frac{5}{8}$ inches instead of 2 inches. This iron chuck-block lasts as long as the mortar, and more amalgam collects on it. Of the free gold recovered 55 per cent. is caught on this inside plate.

With wooden chuck-blocks the copper is removed when the block is worn out and reset on a new one, or they are scraped very carefully, put aside, melted, and sold.

The reason why the Caledonia mill has amalgamated plates at both the back and front is that the ore milled is not at all oxidized, making it more difficult to extract the gold. The aim, therefore, is to keep the pulp longer in the battery and present a larger surface of copper plate for amalgamation. The copper plate in front is 5 inches broad, and that at the back 8 inches. Both are made of $\frac{3}{16}$ inch copper plate bolted direct to the inside of the mortar. The lower edge of the plates is 9 inches above the bottom of the dies. Of the free gold recovered 60 per cent. is caught in the box.

Screens.—The Father de Smet mill uses No. 30 brass-wire screens, while all the other Homestake mills employ diagonal slot No. 7 Russian iron of No. $24\frac{1}{2}$ American wire gauge, weighing 0·987 lb. per square foot. The slots are $\frac{1}{8}$ inch long, and there are 8 to the inch. The punched surface is 48 inches by 7 inches, with a 1 inch margin, making the screen 50 inches by 9 inches. A screen lasts two weeks. The wooden frame is 4 feet 4 inches long by $11\frac{1}{2}$ inches deep outside, and has a strengthening piece down the centre. To fasten the screen in place, the lap is first tacked on to hold it in place, then a strip of rubber-cloth 2 inches wide is placed over it. Small holes are punched through the rubber and the lap of the screen, and both are nailed to the wooden frame, the burr facing the inside. The outside of the frame is protected by

three iron facings, ½ inch by 9 inches by $\frac{3}{16}$ inch thick, fastened at the middle and ends with a couple of wood screws. Screens of aluminium bronze have been tried, and found so satisfactory that they are likely to replace Russian iron entirely.

The Caledonia mill uses No. 24 brass-wire screens, the thickness of the wire being No. 26, and the screen-surface, 48 inches by 5⅜ inches. The screen lasts one week. It is fastened to a plain wooden frame, 53 inches by 12½ inches, the horizontal sides being 3½ inches wide, and the vertical sides 2½ inches. Three wooden ribs, 1 inch wide, divide the surface into four panels, and keep it from bulging outwards. The attachments of the screen and frame are the same as at the Homestake mill, except that here it is keyed against the lower edge of the discharge-opening instead of resting on the board of the chuck-block. The Caledonia mill uses wire screens because, though its stamps drop 3 inches farther than the Homestake mill, the splash is not so great, owing to the greater width of the mortar.

The force of the splash in the narrow Homestake mortar is thrown entirely against the screen, while in the wider Caledonia type it is divided between the screen in front and the recess at the back, hence the slot-screen would clog. The upper part of the discharge of both classes of mortar above the screen-frames is closed either by a 1 inch board or a canvas curtain, or by a piece of old belting suspended from a lath. This hangs down and meets the screen inside the mortar. The curtain has the advantage over the board that the amalgamator can easily pass his hand inside, and remove chips of wood liable to choke the screen. In order to break the fall of the pulp (forcibly driven against the screen) on the apron-plate of the Homestake mill, a splashboard is fastened to the frame of the latter, to prevent any amalgam collected there being washed away. The Caledonia mortar has no splashboard, as the pulp does not pass the screen with sufficient force to endanger the amalgam at the head of the apron-plate outside.

Stamps.—The stamps weigh 850 lbs., and have about 16 lbs. to the square inch of crushing surface. The stem of wrought-iron tapers 6 inches at both ends, so that it can be reversed if broken. A stem lasts about three years at the Homestake mill, before new ends have to be welded on; it is 14 feet long, 3⅛ inches in diameter, and weighs 340 lbs. The cast-iron head is 18 inches high, 9 inches in diameter at the top, 8 inches at the bottom, and weighs 240 lbs.; it is not, as is often the case, fortified with wrought-iron rings. The usual keyways for the

removal of stem and shoe are parallel. To fasten on the head to the stem, the latter is let down through the guideholes and the socket of the head is placed beneath it. The stem is then lifted and dropped, and, if necessary, driven on with a sledge-hammer. Then the stem and head are lifted, and dropped several times on to a piece of timber until wedged firmly together. A head lasts five years at the Homestake mill.

The shoes are made of white cast-iron. The cylindrical portion is 8 inches high and 8¼ inches in diameter, with a tapering shank 5 inches high, 4½ inches in diameter at the base, and 3⅛ inches at the top. They are chilled for 6½ inches from the face, whilst the remaining 1½ inches and the shank are cast in sand and cooled slowly. They weigh 140 lbs.

To fasten the shoe to the head the shank is surrounded by small wooden wedges tied on with string, the shoe is put in position and the head dropped several times on to the dies, which are protected by a piece of planking laid across them.

At the Golden Star mill, instead of tying on the wooden wedges, a strip of canvas is wound round them and tacked to each wedge, forming a sort of collar. This can be slipped over the shank of a new shoe, saving time and labour on clean-up days when shoes are replaced,

After some time a shoe becomes slightly concave, but on the whole wears more evenly than the die.

At the Homestake mill a shoe lasts two months, crushing 270 tons of rock. It is then worn down to 2 inches from the shank and weighs 40 lbs., corresponding to 37 lbs. of iron worn away for every 100 tons of rock crushed.

At the Caledonia mill, a shoe lasts three months and crushes 300 tons. It is replaced when worn down to 1 inch and weighs 35 lbs., corresponding to a consumption of 35 lbs. of iron for every 100 tons of crushed ore.

Tappets.—Gib tappets are used, secured in the Homestake mill with two keys, in the Caledonia with three; they are made of cast-iron, and weigh 130 lbs. The diameter at the ends is 9¼ inches, in the centre (which is 7 inches long) 6 inches. The wearing faces are 2½ inches thick and are reversible: when both become grooved they are planed off in a lathe and replaced. When worn down 1¾ inches (about once in three years) they are replaced by new ones. They rarely split; case-hardened tappets and cams were tried, but the surfaces cracked, and they did not answer well, which has been the writer's own experience elsewhere with steel-faced cams and tappets. Solid steel tappets and cams have not

been tried. It takes between 6 and 8 hours to change the tappets and cams of one battery. The order of drop at the Homestake is 1, 3, 5, 2, 4, at the Caledonia 1, 3, 5, 2, 4, and 1, 4, 2, 5, 3. The Caledonia, crushing harder rock, has a higher drop, 12 inches as compared with 9 inches at the Homestake mill, and consequently has to run more slowly, 74 in place of 85 drops per minute.

Cams and Cam-shafts.—The cams are double and are made of tough cast-iron. The hub, which is on the off-side of the stem, is cast thick enough to stand the strain, but not otherwise strengthened. At the Homestake mill the working-face is 2 inches wide and $3\frac{1}{2}$ inches deep, and at the Caledonia mill $2\frac{1}{2}$ inches wide and 2 inches deep. The hub of both is $3\frac{1}{2}$ inches thick, and the web, which strengthens the cam (commencing deep at the hub, and ending thin at the toe), is $9\frac{1}{2}$ inches deep at the centre in the Homestake pattern and $10\frac{1}{2}$ inches deep in the Caledonia, whilst the distance from the centre of the cam to its point in one case is 17 inches and in the other 19 inches. The cams are made of car-wheel iron and last over four years. They are lubricated with axle grease; and to prevent any of this dropping on to the apron-plates, a curtain of canvas is stretched on a frame below them, and catches any grease thrown off while they are in motion.

The cam-shafts are of tough wrought-iron turned in a lathe. They have one key seat. The keys are of steel, and hand-fitted; wrought-iron keys soon loose their shape, while machine-fitted keys get loose very easily. It takes 10 hours to replace a broken cam-shaft, supposing the keys are ready prepared and fitted. As the fitting of each key-seat takes an hour or more, a well appointed mill should have on hand a spare cam-shaft (or more, depending on its size and situation), with the necessary cams and keys ready for use.

The Homestake cam-shafts were formerly made $4\frac{1}{2}$ inches wide and $4\frac{3}{4}$ inches in diameter, and lasted about five years; now they are made stronger, running up to $5\frac{3}{8}$ inches, and have stood ten years. The distance between the centres of the cam-shaft and stem is $5\frac{1}{2}$ inches.

The cam-shaft of the Caledonia is $4\frac{1}{1}\frac{3}{6}$ inches in diameter, and its centre is $6\frac{1}{2}$ inches distant from the centre of the stem.

The cam-shaft pulleys are built up of wood, varying in diameter from 6 feet to $7\frac{1}{2}$ feet. When put in place the shaft is revolved, and the face turned off true.

Crushing Capacity.—The Homestake stamp develops 78,030,000 foot-pounds in 24 hours, crushing 1 ton of ore for every 17,340,000 foot-pounds

developed. The Caledonian stamp develops 90,576,000 foot-pounds in 24 hours, crushing 1 ton of ore for every 27,447,272 foot-pounds. Thus, although the efficiency of the Caledonian stamp is greatest, it crushes less ore. This is accounted for (*a*) by the greater hardness of the rock; (*b*) the greater width of mortar at the discharge (16 inches as compared with 13½ inches); and (*c*) the recess for the plate at the back of the mortar. With a lower discharge a greater crushing capacity would be expected, but the above reasons explain why this is not the case in practice. The smallness of the Caledonian screen (258 as compared with 376 square inches) may be assumed to be counterbalanced by the Caledonia using No. 24 wire against Homestake No. 7 slot (corresponding to No. 30 wire).

Apron-plates, Traps, and Sluice-boxes.—The pulp passing through the screens flows in small waves down the apron-plate, and during the interval between these waves any quicksilver, amalgam, or fine gold passing over the amalgamated surface has a chance of settling and adhering to it. The plate consists of a single sheet of copper, the width of the mortar-discharge, fastened with iron screws to a wooden table. Except at the Deadwood and Golden Terra mills, which have plates 12 feet long, all the other Homestake mills have their apron-plates 10 feet in length, covered with $\frac{3}{16}$ inch copper plate, falling 2 inches to the foot, and discharging into a copper-lined sluice leading to the mercury-trap. The Caledonia apron-plates are 8 feet long, 4 feet 8 inches wide, and the copper plate is $\frac{1}{8}$ inch thick, set at the same grade as the other mills.

The wooden table extends 4 feet beyond the end of the copper plate, narrowing to 4 feet. It has a 1 inch rib down the centre; it is overlaid by two blankets, 5 feet wide and 22 inches long, the upper overlapping the lower one. On these the heavy sands collect; they are washed every half-hour, and they last six months. The pulp from the blankets flows into the mercury-traps, one being placed in the middle of the discharge from each plate.

The plates are of Lake Superior copper, furnished ready for use, and do not require to be annealed; they must be flattened, however, with wooden mallets to make them lie flat and remove any inequalities.

At the Homestake mill they are first scoured with sand-paper, followed by emery cloth, or with tailings rubbed on with a wooden block, 4 inches square, until the face is perfectly bright. If necessary, the sand is moistened with a weak solution of cyanide of potassium, and black spots are often removed with dilute nitric acid. The bright surface of the copper then receives a washing with a strong solution of cyanide of

potassium applied with a soft brush. After two days, the mercury is sprinkled over the plate and rubbed into it with a moist cloth and tailings. When the plate is thoroughly amalgamated it is put in position. More than the usual quantity of mercury is added to the box at first, so that the plate may get into proper condition. This takes from two to four weeks, and to dissolve the copper salts, which stain it during this period, cyanide of potassium or ammonia is added to the battery water.

The mercury-traps save amalgam and mercury not caught on the apron-plates. There are also additional traps at the ends of the sluice-plates outside.

The importance of this simple contrivance is shown by the fact that since their introduction 80 ounces of amalgam and 144 ounces of mercury are recovered at the Homestake 80 stamp-mill monthly, by the inside traps; whilst the outside ones collect 10 to 12 ounces of amalgam and 40 ounces of mercury. They are emptied monthly. At the Caledonia mill the traps are emptied daily (when the plates are dressed) on account of the accumulation of pyrites.

The inside traps at the Homestake mills are wooden boxes, 14 inches long, 17 inches wide, and 24 inches deep, with a copper-lined bottom. They contain three sliding wrought-iron plates, parallel with the short sides of the boxes, set $2\frac{1}{2}$ inches apart. The central partition extends to the bottom, and the two others are 3 inches above it. The pulp flows under the first, over the middle, and under the third.

The outside traps are 48 inches long, 14 inches wide, and 48 inches deep, with three partitions, set $10\frac{1}{2}$ inches apart, reaching from the bottom, to within $1\frac{1}{2}$, 3, and 4 inches below the level of the inlet, the outlet being 6 inches lower. In the middle, between two of the wooden partitions, a sliding wrought-iron plate, $\frac{3}{8}$ inch thick, reaches to within 3 inches of the bottom of each box. The Caledonia traps are smaller, as there is one for each apron-plate.

The sluice-boxes, which are below the inside traps, are simple wooden launders, lined on the bottom with copper plates. At the Homestake mill, they are 8 to 10 feet long, 18 inches broad, and have a fall of 1 inch per foot. The copper plate is $\frac{1}{8}$ inch thick. At the Caledonia mill they are 8 feet long and 8 inches broad, as less pulp passes through them.

Labour.—All the heads of the different departments are responsible to the superintendent. The mill proper is under an experienced foreman, one foreman being sometimes in charge of several mills.

Next comes the millwright, who in large mills sometimes has an

assistant, called the pipe-fitter. The millwright combines the trades of carpenter and machinist, making and replacing new guides, exchanging cams and cam-shafts, fastening loose cams, replacing screens, making and repairing chuck-blocks, exchanging shoes and dies of crushers, and looking after the water connexions, etc.—in fact looking after all the mechanical work about the mill.

The machinist is in charge of the repair-shop and is generally under the millwright, though at the Caledonia mill the foreman takes this duty; any extensive repairs are made at the Homestake shops.

As the mills are driven by steam, each has two enginemen responsible for their firemen.

There is a night watchman generally for each mill, to guard against fire or other accident.

The man who has the immediate charge of the running of the mill is the head amalgamator. He, like all the other heads of departments, is directly under the foreman, and is in turn responsible for his assistants, amalgamators, crusher-men, oilers, feeders, and labourers. In addition to running the mill he has charge of the collection and keeping of the amalgam, and must therefore be not only capable but trustworthy.

The amalgamators feed quicksilver, regulate the water-supply, and look after the running of the battery in general. Quicksilver is fed every half hour with a wooden spoon like a mustard spoon. The quantity used every 24 hours varies from ¼ to ½ lb. for each battery according to the character of the ore.* The correct amount is determined by the feel of the amalgam collected on the plates. If hard and crumbly there is danger of its being carried off by the pulp, and more quicksilver must be added. On the other hand, too much quicksilver makes the outside copper plates too soft and slippery, with the risk of liquid amalgam rolling off, while less amalgam collects on the inner ones.

All the quicksilver is added to the mortar in the Homestake mills, and the amalgam is of medium hardness. At the Caledonia mill the aim is, by adding part to the mortar and the rest to the apron-plates, to make the inside amalgam as hard as may be, and to keep the amalgam on the aprons softer than on those of the Homestake mills.

Each management is satisfied with its own method, and, perhaps, the gold of the Caledonia ore being coarser than the Homestake mill may justify the difference of method.

* Fine gold requires more mercury in amalgamating the same weight of gold, and the loss of mercury is liable to be somewhat increased. Great losses may be occasioned by ores containing heavy sulphides or by over-handling.

The loss of quicksilver at the Homestake mills per year per stamp is 5·27 lbs., or 0·0044 lb. per ton of rock crushed. The loss at the Caledonia mills is 7 lbs. per year per stamp, or 0·0011 lb. per ton of stone crushed.* With the harder and more pyritic ore of the Caledonia mine more mercury is liable to be floured per stamp, but owing to the smaller quantity of rock crushed per stamp, less quicksilver is lost per ton.

The Homestake mills use 1 miner's inch of water per battery, and the Caledonia mill 1¼ inches.

To set the tappets, which is very necessary to maintain the height of drop constant, whether the shoes be new or worn, the stamps are hung up, the mortar opened, the stamps lifted by an iron bar, and a block of wood, 1 inch higher than the desired drop, is placed between shoe and die. The tappet is then loosened, allowed to fall on the prop, and again keyed fast. As the point of the prop (finger), and the blocks used to support the shoe are both 1 inch higher than the required drop, on removing the block, the stamps being each in turn regulated thus, will have the desired uniformity of drop, while the different levels at which the tappets are keyed to the stem, will indicate how much the shoe and die are worn down.

The crusher-men, in addition to tending the grizzlies, breaking the coarse lumps, and feeding the crushers, have to watch for and take out any pieces of wood or iron found in the ore, and throw them aside.

All small pieces of wood finding their way into the mortar are removed by the amalgamators, but very little ought to escape the notice of the crusher-men and ore-feeder men, who should remove any pieces they notice, from the shoots of the automatic feeders.

The oilers have to keep all working parts of the machinery properly lubricated, and should be especially careful to guard against excess of grease about the battery.

The feeders attend to the regular and uniform feeding of the ore; the height of ore between shoe and die should never be more than 1 inch, and as much less as possible without allowing the stamps to pound.

One or two labourers are generally needed to do extra work, which does not fall into the usual routine.

The shifts in the mills are changed monthly.

Only three more men are required in the Golden Star 120 stamp mill, than are employed in the 80 stamp mill of the Homestake works running

* In most cases dealing with free-milling ore, ½ to ¾ oz. per ton of ore milled, or 12 to 15 lbs. per month, is the loss to be expected in, say, a 20 stamp mill.

on the same ore. From this it will be seen that a large number of stamps, whilst greatly increasing production, does not proportionately increase the labour outlay.

Collection of Amalgam, and Dressing the Plates.—The amalgam which has collected on the apron-plates the previous day is removed every morning with the change of night shift. An amalgamator, each with an assistant, has charge of this work. The method at the Golden Star mill will serve to illustrate how this work is done. When the copper plates are to be cleaned the stamps are hung up, the water is turned off, and the splashboard removed, and washed at the head of the apron-plates with a hose. It is then placed at the lower end of the plate, and the hose is turned on the screen and apron to remove any sand collected on them. The copper plate should now be clear and bright, or silver white where the amalgam has collected, though here and there spots may be left on it, which are generally at first a light yellow, but turn darker with exposure to air. The plates may be scraped with a blunt double-edged chisel. Then two men loosen the amalgam with heavy whisk-brushes, beginning at the top and working downwards. When this is done, the amalgam is swept in the opposite direction, and collected at the head of the apron. There it is brushed into an amalgam scoop with a rubber scraper (a sharp-edged piece of belting) and emptied into a small enamelled-iron dish. After this the plates are brightened by brushing them with a whisk-broom, using tailings moistened with dilute cyanide of potassium solution, the men working from the head of the plate downwards. If necessary, a little mercury is sprinkled on to the plate, from a bottle over the neck of which a piece of canvas is stretched and tied. After being cleaned, the plates are smoothed with soft paintbrushes passed transversely over them, beginning at the bottom. This finishes the operation, which lasts 4 hours for 24 plates, or 10 minutes for each battery. The indiscriminate use of acids or alkalies on the plates is strongly to be condemned, as they tend gradually to alter the nature of the copper, and if applied in excess precipitate verdigris in a few days, or form salts of copper, which, becoming gradually converted into oxides, give additional trouble. The whisk-brooms used for brushing are of the ordinary kind, cut short to stiffen them. The brushing should be done in straight lines, commencing at the top, the amalgam being generally brushed back to the top, but sometimes it is removed at the bottom. It should not be brushed towards the centre.

If plates are not run too wet—*i.e.*, with an excess of mercury, the

chances of oxidation are reduced by this method of procedure to a minimum, and a thin coating of amalgam is left over the entire surface, excluding air and preventing verdigris.

The amalgam obtained is contaminated with impurities. To remove these, it is placed in a mortar and diluted with mercury. The amalgamator then adds water, and grinds the amalgam so as to bring all the impurities to the surface. These may be in part washed off (the sands) with a hose, and in part removed with a sponge or wet cloth, which takes up the base-metal amalgam, until the surface of the mercury in the mortar is bright and clear like a mirror. It is then passed through a small strainer, and the residual pasty amalgam is transferred to a piece of linen, and the excess of quicksilver is expressed by wringing. The ball of hard amalgam is locked up in the safe, and kept till the next clean-up. All the sands are returned to the battery, and the quicksilver goes back into stock.

Clean-up.—Twice a month the gold amalgam adhering to the inner copper plates is removed, and any repairs needed about the mill are made. At the Caledonia mill the bi-monthly clean-ups are similar to that at the Homestake mills on the first day of the month. At the last named mills the semi-monthly clean-up is different. On the first day of the month the entire mortar is emptied, and shoes and dies are changed if need be; while on the 15th only the amalgam from the inside plates is removed, and the mercury-traps emptied.

At the Golden Star mill the clean-up on the first day of the month is carried out as follows:—It begins at 7 a.m., the feeding of the battery is stopped a quarter of an hour previously, and the stamps are made to drop slowly, so that at 7 a.m. no more ore may be left in the mortar above the screen-frame. The splashboards are removed, the stamps hung up, the water shut off, and the engine stopped. The mortars on one side of the mill are then opened by removing the curtains, screens, and chuck-blocks. The curtains and screens are first roughly washed by playing the hose over them, and put aside to be more carefully cleaned later. The six chuck-blocks from the batteries on the side of the mill being cleaned, are placed on two apron-plates, at each of which there are four men stationed to remove the amalgam, under the supervision of the head amalgamator. This is done by scraping the copper plates with a chisel when the hard amalgam drops off on to the apron-plate beneath, as much amalgam as possible being removed without exposing the copper; quicksilver is then sprinkled over the plate (to dilute the hard amalgam), then spread evenly over the plate, brightened by scouring with a whisk-broom and tailings,

and finally smoothed with a soft paintbrush. The amalgam that has dropped on to the apron-plate from the three chuck-blocks is collected at the head of the former, and put under lock and key. Thus the chuck-blocks of the entire mill are scraped and cleaned in four sets of six each.

In the meantime, another set of men scrape and wash the rim and flanges of the mortar and collect the amalgam. They also remove the apron-plate amalgam which has accumulated during the previous day. In order to keep the apron-plates soft a little quicksilver is sprinkled over them and evenly distributed with a brush, but they are not dressed till later.

As soon as the amalgam from the apron-plates has been removed, two small platforms are placed across the head of the table, in front of the mortar, for the men to stand on. They then bale out the water still remaining in the bottom, and shovel out the sands above the dies into a heap on the apron-plate (more usually removed in buckets and collected in a tub at the side). These sands are returned to the battery after the dies have been put in place, as they do not contain amalgam. Before the die can be taken out, the stamp has to be raised higher; to do this a block and tackle were formerly used, now a piece of timber is placed crossways on the rests of the splashboard, serving as a fulcrum for an iron bar, with which the head is lifted. It is kept in position by a 4 inches piece of wood on the prop (finger) of the stamp, on which the tappet is let down. The dies are prized up with the bar, lifted out, and roughly cleaned. Those to be exchanged are taken away, and piled up to be carefully scraped and washed in due time. Those that are still fit for use are returned to the mortar, after they have been scrubbed with a scrubbing-brush in a tub.

After the dies have been taken out, the remaining sand is shovelled out and piled in a separate tub to be treated afterwards in the rocker and pan. It is rich in amalgam and contains bits of iron, etc. Any particles of amalgam that may have adhered to the rough sides of the mortar are washed down and added to the sands. The dies are now replaced, new shoes, if required, are put in place on the dies, and the wooden collar slipped over the shank. Then the recesses of the chuck-block, screen-frame, etc., are cleaned in a tub by playing a hose on them, after which they too are put in.

When the chuck-block is in place, and the screens have been scrubbed down with a brush and water, the sands first taken out are shovelled back to fill the bottom of the mortar up to the top of the dies, and the drop of the stamps is next regulated.

If new shoes have to be substituted for old ones, they are fixed on by letting the heads drop on to them, as previously described; the wooden block, 1 inch higher than the drop, is placed across the dies, and each stamp in turn is let down till the head rests on the block, the keys of the tappets are loosened, allowing them to fall on the props, and they are then keyed up again. The screens are then replaced, the apron-plates being dressed in the usual way; any amalgam clinging to the small sluices leading to the traps and to the sluice-boxes is removed, and these are dressed like the apron-plates.

The splashboards are put back in place, some ore is fed into the mortar, the water is turned on, and the stamps of one battery after another are let down from the props, the engine running slowly at first, and, as the last head falls, gradually picking up speed, till the regular beat of the heads reverberates along the line, a sound as familiar as it is inspiriting to dwellers in mining camps.

As the mortars are empty when the mill starts up, care must be taken to regulate the ore-supply accordingly.

In cleaning up a mill all hands have to take part in it, the night shift working 6 hours extra.

This description of the clean-up of a 120 stamp mill shows how it is possible to accomplish in the short space of 7 hours (without outside help) what formerly used to take a day, through the various operations being systematized and worked into one another. When the clean-up of the mill is over and the stamps are once more running, the sands shovelled out from the bottom of the mortars have to be worked up and the amalgam cleaned for retorting.

Two crusher-men are detailed to clean the sands, which are first washed in a rocker. Any coarse bits of iron are picked out and collected in a dish. When the sands have been rocked for sometime, and the hose played on them, the residue remaining in the hopper of the rocker is broken as fine as possible with a wooden mallet. The coarse particles remaining are washed in a coarse screen over the clean-up pan, any amalgam remaining on the screen being picked out and put into the pan. The sands go back to the battery, and the sulphides, etc., which collect on the curtain and riffle of the rocker, are taken out and put into the pan. The settlings in the sluice which conducts the slimes to the waste-flume are shovelled out, and returned to the battery.

To clean the amalgam collected from chuck-blocks, apron-plates, sluices, mortars, shoes, dies, screens, etc., it is charged with water into the clean-up pan (5 feet in diameter, the muller making 30 revolutions

per minute), and from 600 to 700 lbs. of quicksilver is added.* It takes about three hours to clean in the pan all the by-products containing amalgam. When this is all collected and the water above is fairly clear, the muller is raised with block and tackle, and the entire contents of the pan are emptied through the lowest discharge-opening into a square box which overflows into the tailings discharge-box. The muller and the bottom of the pan are well brushed out, with a stream of water flowing in all the time, and the liquid amalgam in the box is drained of water and passed through a strainer. The pasty amalgam is removed and freed of the excess of quicksilver by wringing it in canvas bags under water. The balls of hard amalgam resulting contain about 38 per cent. of gold. The mercury collected below the strainer goes back to the main stock;† that squeezed from the pasty amalgam is first purified by adding some nitric acid, stirring it, and washing with water.

The semi-monthly clean up is much simpler, only the chuck-blocks are taken out and cleaned, replacing dies and shoes if necessary, and cleaning the traps. Their contents go to the pan and are worked with the other products containing amalgam. This clean-up lasts 5 hours.

Once a year the old iron and wood chips collected during the previous 12 months are worked over. The pieces of iron, which are scraped to remove any bits of amalgam adhering to them when they are picked out from the battery-bottoms, are thrown into a heap in the yard, and left to corrode by atmospheric agency. Oxidation is hastened by adding salt to the heap at intervals. The iron at the end of the year having fallen to pieces, is charged with mercury into the pans and its gold extracted. The chips of wood picked out of the mortar are likewise collected in a box, and are once a year burnt in a heap in the yard, and the ashes are collected and amalgamated in the pan. In this way 16 to 18 lbs. of amalgam is saved every year from the two mills of the Homestake Company containing 200 stamps.

Retorting and Melting.—To remove the mercury from the hard amalgam in balls, cylindrical or bulb retorts are used. The cylindrical retort of the Homestake Mill Company is 12 inches in diameter and 3 feet long, holding 1,000 lbs. of amalgam. The usual charge of 500 lbs. is retorted in about 6 hours, using $\frac{1}{4}$ cord of wood. The retort metal (crude bullion) amounts to 38 or 40 per cent. of the amalgam in the

* Under ordinary circumstances in most mills 200 lbs. is sufficient.

† A large mill of, say, 80 stamps will require about 6 flasks in stock; a 10 stamp mill 3.

charge. At the Caledonia mill it is only 33 per cent.; this no doubt is due to the amalgam being less tightly squeezed. At the Deadwood Terra mill it is often only 25 per cent., and this is accounted for by the fine condition of the gold.

The crude-bullion is melted into bars, using, at the Homestake mill, the 1,500 ounce silver mould (5 inches by 5 inches by 11½ inches), or the 700 ounce mould (3½ inches by 4 inches by 9½ inches). The bars are cast 3 to 4 inches thick, and weigh 1,000 to 1,400 ounces. It takes four hours to melt four 1,400 ounce bars, and the crucible lasts for 8 to 12 charges. The moulds must be warmed and smoked inside before pouring. A little borax or bi-carbonate of soda is invariably added to the melt. If iron be present a little nitrate of potassium is used. Bone-ash is used to thicken the slag and make it skim easily. Phosphorus (in quantities not exceeding half an ounce per melt of 1,800 ounces) may be employed to get rid of the copper, and corrosive sublimate added for bullion containing much lead or antimony.

The loss in melting Homestake bullion is 1·5 per cent., and the average composition of the bars is 820 parts of gold, 165 parts of silver, and 15 parts of base metal. The loss on Caledonia bullion is greater, being 7 per cent., as the amalgam is less carefully cleaned. The average composition of the bullion is: gold, 798 parts; silver, 182 parts; and base metal, 20 parts. The bullion is sampled (chipped or drillings taken), weighed, assayed, and shipped.

Geology of the District.

In the neighbourhood of the Homestake group of mines numerous sheets of porphyry (or more properly speaking felsite) are met with, sometimes cutting across the stratification of the country, but more frequently parallel with it. In the northern half of the belt—the Deadwood Terra and De Smet end—the surface was once overlain by felsite which appears to have been injected between the slates and Potsdam formation of the district, and can still be seen capping the ridges between Gold Run and Bobtail Gulch, and Bobtail Gulch and Deadwood Creek.

Mining.

The cost of operations at the Homestake mine for the year ending June 1st, 1888, was as follows:—Mining, 7s.; milling, 3s. 5½d.; total, 10s. 5½d. per ton; leaving a profit of 4s. 11½d. per ton on 15s. 5½d. ore.

The mining costs were sub-divided as follows:—

	s.	d.
Labour	4	6·50
Dead work	0	11·35
Supplies	0	2·76
Powder	0	0·55
Candles	0	0·84
Machinery	0	2·18
Oil	0	0·64
Timber	0	8·63
Wood	0	2·52
Coal	0	0·21
Total	7	0·18

The cost of mining and milling at the Deadwood Terra mine are stated to be at the present time (1892) 5s. 2½d.* Chlorination-works have of recent years been established in Dakota.

Working Results.

Between June, 1887, and June, 1888, the yield in free gold from the Homestake mines represented 15s. 4d. per ton. Assuming that 85 per cent. of the free gold is saved, the ore would in that case run 18s. 0½d. in value in free gold per ton. Its total value varies from £1 0s. 10d. to £2 1s. 8d. per ton, whilst the amount of concentrates does not exceed 3 per cent. Their value, as shown by experiments, is £5 a ton, the average assay of the tailings is estimated at 6s. 3d. per ton.

Two sets of experiments were made in the spring of 1885 on the Homestake and Golden Star tailings, before and after the introduction of mercury-traps. Before they were introduced 1,124 tons of concentrates (blanketings) had been collected in a separate building (the blanket-house), which assayed £7 6s. 6d. per ton. These, panned down, gave 20·5 per cent. of cleaner concentrates, assaying £8 7s. 5d., and a second grade assaying £3 11s. 2d. per ton. The former yielded by pan amalgamation 56·9 per cent. of their total value.

When, in consequence of these tests the mercury-traps were introduced, the loss was reduced; the concentrates then saved assayed £5 15s. 1½d. per ton, and yielded 92 per cent. of their gold in the pans, but the pyrites still assayed £2 11s. 1d. per ton. The concentration by blankets being too expensive, it was given up.

The tailings from the Highland ore average 4s. 2d. per ton, and from the Deadwood Terra 2s. 1d., seldom running above 3s. 1½d. per ton.

In regard to the fineness to which it is necessary to crush the ore, Dr. Goering made a number of tests to find the relation between size and

* *The Engineering and Mining Journal*, (New York), vol. lv., page 338.

assay value of tailings, samples being taken hourly and the sands obtained dried, weighed, and screened through different sieves. The results were :—

Per Cent. in Weight.	Passing Through Screen. No.	Remaining on Screen. No.	Assay Value per Ton. s. d.
94·07	50	—	5 2½
2·53	50	40	8 10½
3·40	40	—	11 6

These results show that the loss in the tailings increases rapidly if the screens be allowed to remain too long in use. Another set of experiments on tailings running 8s. 4d. per ton, screened through a No. 30 screen, showed that 6 per cent., which did not pass through the screen, assayed as high as £1 0s. 11d. per ton. The result proved that the heavy Russian iron screens should be changed fortnightly.

The Caledonia mill crushed from May, 1887, to May, 1888, 73,425 tons of stone, yielding bullion equivalent to a return of 16s. 9d. per ton in free gold. The blanket concentrates (amalgamated raw in pans) yielded pure pyrites assaying £18 15s. per ton, and the tailings from the blankets when panned gave concentrates worth £1 9s. 2d. to £1 17s. 6d. per ton.

The cost of milling at the Caledonia, a 60-stamp mill, in 1887-88 was 3s. 7½d. per ton, one-third chargeable to labour and two-thirds to material ; a low figure, considering the rock is hard, compared with the Homestake ore.

The two striking features of the Dakota practice are the cheapness, simplicity, and effectiveness of the method by which the free gold is extracted, and the waste of sulphides in the tailings.

Mr. Hofman suggests that in view of the fact that the sulphides would appear to average 3 per cent., worth £5 per ton, they might be advantageously dealt with by passing the pulp through *spitz-lütten* to sort out the coarse sands and mineral, the overflow of the *spitz-lütten* going on to a series of *spitz-kasten*, the outflow of which would be waste. The coarse products drawn from the *spitz-lütten* would contain, according to the experiments made in 1885, free gold, which could be recovered by crushing wet in Chilian mills or rolls, and allowing the pulp to flow over amalgamated plates, and then to pass over a separate series of classifiers (*spitz-kasten*), or go back to the main system. The graded pulp obtained by the different *spitz-kasten* would be separated on round tables into concentrates, middlings, or waste. The middling would be re-worked or pumped back to the main system of *spitz-kasten*.

The cost of concentrating the tailings in this manner would probably

not exceed 4s. 2d. per ton of sulphides. The concentrates could be worked by barrel-chlorination, the total cost of which Mr. Hofman states would probably not exceed £1 13s. 4d. per ton of concentrates. He recommends a combination of two systems of furnaces as likely to cheapen the cost of roasting (where wood costs in this district £1 5s. per cord and labour 12s. 6d. to 14s. 7d. a day). The Spence automatic furnace would do the preliminary roasting cheaply, and the revolving-hearth (the Brunton) would effectually dead roast large quantities of ore, the sulphur of which had been nearly all eliminated. The modern modifications of the latter furnace, such as the Pearce turret or Blake circular hearth, are, the writer believes, specially adapted for dealing with fine ore liable to dust.

The Golden Reward Chlorination Works.

The cost of treating gold ores by barrel-chlorination on a large scale, at the Golden Reward mill, Deadwood, Dakota, has been obtained from careful daily records of the ore treated per 24 hours: these were averaged semi-monthly and monthly, to see where improvements and reductions could be made.* For the months of July, August, September, and part of October, 1891, the amount of ore treated and cost of treatment (including all working expenses, inclusive of interest on capital, taxes, insurance, etc., which always vary with the financial management of works of this character) were as follows:—

1891.	July. Tons.	Aug. Tons.	September. Tons.	First Half of October. Tons.
Amount of ore treated ...	1,430	1,194·75	1,512·75	871
Average amount per day ...	46·13	38·54	50·42	54·43
Costs per ton—	s. d.	s. d.	s. d.	s. d.
Milling	6 2½	5 10	6 0	4 11½
Roasting	6 4½	5 7½	5 8½	5 5
Chlorination	7 4	6 11½	6 10½	6 5½
Office salaries	1 8	1 11½	1 6½	1 4
Construction and repairs	1 6	3 10½	0 10½	1 4½
Total cost per ton	23 1	24 3	21 0	19 6½

In the month of August the works were closed down for one week, while building a dust-chamber and flue to the roasting-furnaces, which accounts for the higher cost per ton and lower daily average of ore treated, also for the large item for construction and repairs. It will be seen that the cost of milling is excessive. This was due, principally, to the very

* *The Engineering and Mining Journal* (New York), vol. lv., page 269.

inconvenient arrangement of the mill requiring many more men to handle the ore than would be necessary if the machinery were arranged differently. The roasting was done in Brückner cylinder-furnaces holding 3 tons to a charge; since then, a revolving continuous feed-and-discharge roasting-cylinder, arranged with self-feeding dust-chamber, has been added, which has brought the cost below 4s. 2d. per ton.

The amount of chemicals used in the chlorination and precipitation departments to treat the ore milled in each month, as above, was as follows:—

	July. Consumption		August. Consumption.		September. Consumption.		First Half Oct. Consumption.	
	Total.	Per Ton.	Total.	Per Ton.	Total.	Per Ton.	Total.	Per Ton.
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.
Sulphuric acid ...	37,264	26·82	30,044	25·14	38,083	25·17	21,580	24·77
Chloride of lime ...	15,790	11·04	12,160	10·17	15,840	10·47	8·790	10·09
Crude sulphur ...	542	0·37	381	0·31	479	0·31	253	0·29
Iron sulphide ...	1,106	0·77	952	0·79	1,263	0·82	736	0·84
Costs per ton of ore treated—		s. d.		s. d.		s. d.		s. d.
Sulphuric acid ...	—	1 10¾	—	1 10	—	1 10	—	1 9½
Chloride of lime...	—	1 5¾	—	1 4½	—	1 5	—	1 4⅜
Crude sulphur ...	—	0 0¾	—	0 0½	—	0 0¾	—	0 0¾
Iron sulphide ...	—	0 2¼	—	0 2	—	0 2	—	0 2⅛
Total cost of chemicals ...	—	3 7¼	—	3 5	—	3 5¾	—	3 4¾

Mr. Rothwell considers it possible to treat gold ores, similar to those treated in the Golden Reward mill, for from 8s. 4d. to 10s. 5d. per ton by barrel-chlorination, with a mill constructed on the latest improved plans for economical work, and having a capacity of 125 to 200 tons per day.

In the eastern and southern districts of the United States, where labour, fuel, and supplies cost about one-half of what they do in the western, the cost of treatment ought to be considerably less—with such a plant.

On some exceptional gold ores that do not require roasting, and that are not adapted to free-milling, it is possible for chlorination to compete favourably with amalgamation as usually practised in the stamp-mill, by saving a higher percentage of gold. Thus, in chlorination the fine or float-gold will be saved, and the coarser particles having been brightened by attrition and chemical action (by sluicing the leached pulp over copper plates), the gold in this condition, could be readily amalgamated and saved.

A general outline of the Dakota process has already been given,* but the following additional details, embodying the latest improvements in the method as described by Mr. Rothwell†, will be of interest.

The mill, roaster, chlorination, and power buildings are erected on level ground, and the main ore-bins, larger crusher, and dryer on benches cut in the hill side. The chief objection to the Plattner process, for a plant of 50 tons or more capacity per 24 hours, is its enormous size, and the length of time that it takes to complete a single operation. The problem, therefore, that the engineer has to solve in attempting to treat low-grade ores which will not concentrate is to find a process that will treat his ores in large quantity expeditiously, cheaply, and with as little interruption as possible. It is claimed that these objects are attained by the arrangements in question.

The ore from the mine comes in cars to the top of the main ore-bin, which has a capacity of about 1,000 tons. The ore from the bin goes over a grizzly (with the bars set 1½ inches apart) which delivers the lumps to a rock-breaker that breaks it to 1½ inches size. The product of the breaker, joining the fines which have passed the grizzly, is fed to a revolving dryer, which has a cylindrical shell built of ⅜ inch tank steel, 5 feet in diameter and 18 feet long, set on two heavy cast-iron tyres 4 feet from each end. These tyres rest on adjustable flanged rollers; the roller-frames are bolted to a heavy timber framework, to one end of which is fastened a pivotal casting, while the other end is provided with a set of screw-jacks to permit of the inclination of the cylinder being changed.

The cylinder has usually an inclination of ¾ to 1 inch per foot, and is revolved by spur-gearing round its exterior with an intermediate gearing of cone-pulleys and friction-clutch, which gives a range of speed of ¾ to 2 revolutions per minute. The friction-clutch allows the dryer to be stopped without stopping the mill, and is much simpler than a fast-and-loose pulley and shifting belt. To increase its drying capacity, the cylinder is divided into four longitudinal compartments by double iron-plates ⅜ inch thick, bolted to angle-irons riveted to the shell, and an X iron in the centre of the cylinder. The compartments have a length of 14 feet, which allows 2 feet of free space at each end of the cylinder.

The furnace is built of masonry well strapped with iron, and with ample air-passages around the fireplace, and in the bridge-wall, through which a large volume of air can be passed and heated before entering the cylinder.

* *Trans. Fed. Inst.*, vol. iv., page 249.

† *The Mineral Industry*, 1892, page 233.

The dust-chamber is similar, but not quite so large as that of the roaster, to be described later. From the dryer the ore passes by gravity to the fine crusher (rock-breaker) where it is crushed to pass through $\frac{3}{4}$ inch mesh, and is then raised by a chain-elevator to the first screen. This is double, with a coarse mesh within a fine mesh, the object being to protect the latter from undue wear. The screen is hexagonal in form, 9 feet long, 4 feet 6 inches outside diameter at one end, and 5 feet 6 inches diameter at the other. The inner screen is 12 inches less in diameter at each end.

The frame of the screen is a cast-iron hub with six radial arms of 1 inch round iron reduced to $\frac{3}{4}$ inch near the ends. Two of these hubs are keyed on the $\frac{3}{4}$ inch shaft, together with the cast-iron headpiece.

The mesh frames are slid in place in grooved cast-iron pieces fastened to the radial arms of the screen-frame. They are interchangeable, and there are always three spare ones on hand.

The screens make $8\frac{1}{2}$ revolutions per minute. The outside casing is made of double thickness, plain timber inside, and groove and tongue outside, with paper between. The doors, one at each side and one at the end, are hinged to the frame of the casing.

The ore that passes the finer mesh goes to an elevator, and is discharged into the storage-bins; the rejected ore goes through a shoot to a large hopper over the coarse rolls (26 by 15 inches, with heavy steel tyres) which are driven by belts running at 90 revolutions per minute.

The ore from the rolls is elevated to the main sizing-screens, of which there are two of the same construction as the one already described, except that they are only 12 feet long, and are driven with separate bevel gearing and clutch to each screen, so that in case of need one can be stopped, and then the whole quantity of ore passed over the other. The fine crushed ore goes to storage-bins and the coarse to fine crushing-rolls, which are the same size as the coarse ones, and is returned again to the screens.

The rolls are fed by automatic feeders, while an exhaust fan draws off the dust, and discharges it into a collector on the top of the ore-bins. The storage-bins are placed along one side of the mill-building, and have a capacity of about 200 tons of crushed ore. A conveyor carries the ore from the bins to the roasting-cylinder. The latter is 36 feet long, 5 feet in diameter, and built of $\frac{1}{3}$ inch plate. It revolves on five tyres which, like the rollers, spur, pinion, and other gear, correspond with those of the drying cylinder, so that it is only necessary to keep one set of extra parts. The shell is lined throughout with fireclay blocks 6 inches thick, and

moulded to fit the circle. It has projecting shelves the whole length of the cylinder, which raise the ore and shower it through the hot oxidizing gases in the upper part of the cylinder. At the end exposed to the fire, the shell is further protected by a specially shaped block which overlaps the end of the plate, and is held in place by a small piece of square bar iron (fastened to the shell) which fits into a groove in the block. The inclination of the cylinder is 14 inches, and it revolves once per minute.

The dust-chamber, which is arranged to feed the dust back into the roasting-cylinder is hopper-shaped on three sides, the bottom being an inclined cast-iron plate projecting about 8 inches into the upper end of the cylinder. The dust carried out of the cylinder settles in this chamber and, as it accumulates, slides down and mixes with the fresh ore. This arrangement does away with the old auxiliary fireplace and the rehandling of the dust, besides giving the ore to the chlorinators in a much more uniform condition than when the dust is mixed with the ore by hand. From the dust-chamber, the gases pass up an inclined flue on the hillside to a chimney 42 inches in diameter and 60 feet high.

The furnace fireplace is constructed with air-channels and openings through which the temperature and working of the cylinder can be regulated and watched. The fire-arch and bridge-wall are so built that the flame is directed into the lower part of the cylinder, and against the ore as it slides down on the lining. The roasted ore is discharged into a hopper, from which it is drawn into cars and spread on the cooling-floor.

To cool the ore, it is spread out thinly on the floor and furrowed. When the surface has partially cooled water is sprinkled over it, and the whole is raked over again; when sufficiently cool, it is sent to an elevator that discharges into hoppers over the chlorination-barrels. These hoppers are made of No. 14 sheet-iron and have a capacity of between 5 and 6 tons each.

The two chlorination-barrels each have a capacity of 5 tons per charge or 35 to 40 tons per 24 hours. The shells are of tank-steel, $\frac{1}{2}$ inch thick, 9 feet long, and 5 feet in diameter inside, whilst the heads are of cast-iron inserted into the end of the shell and bolted through flange and shell. They are $2\frac{1}{2}$ inches thick and heavily ribbed. The trunnions, which are a part of the head, are 12 inches in diameter and 12 inches long where the bearing comes, and 14 inches in diameter and $4\frac{1}{2}$ inches long inside the bearing, making a total length of $16\frac{1}{2}$ inches; they are bored to fit a 3 inches bolt which passes through them and the entire length of the barrel.

Each barrel has two charging-holes (11 inches by 16 inches) oval in shape. The covers are of cast-iron, and when in place are held down by two heavy yokes and four 1½ inches bolts. An eye-bolt is screwed to the centre of each cover to lift it off, and for this purpose a swinging lever is used which holds it out of the way while the barrel is being charged.

The barrels are lined with lead ½ inch in thickness on the heads, and weighing 24 lbs. per square foot, while the shell and other parts exposed to gas or solution are covered with lead ⅜ inch in thickness, and weighing 18 lbs. per square foot. The steel shell is made in one sheet with butt joint and cover-plate, and all the rivet heads are countersunk, so that the inside is perfectly smooth. The lead lining is bolted on with flat-headed lead-covered bolts, which prevents any solution getting between the lining and the shell. The barrel is driven with spur-gearing encircling the shell by a pinion and friction-wheel, which can be thrown in to or out of gear, and a brake is also arranged so that the barrel can be stopped at any point in its revolution.

The supporting diaphragm for the filter is placed so that it will assume a horizontal position when the charging-holes are in place for charging. This diaphragm has an area of nearly 30 square feet, being 8 feet 2 inches long by 3 feet 8 inches wide, and is put into the barrel in the following manner :—Two strips of wood, 2½ inches thick on one edge and 1½ inches on the other, 6 inches wide, and the length of the barrel inside, are bolted to the shell through holes left for that purpose. Below these strips is built a lining of wooden staves, 1½ inches thick and 5 inches wide, on which are placed the supporting segments 3 inches thick. One is placed against each head, and five others spaced equidistant between, which brings them nearly 13 inches apart. On the top of the segments the corrugated plates are laid lengthways with the barrel. These plates are of wood, 2 inches thick, four of which in width form the filtering surface. Each plate is corrugated lengthways with grooves $\frac{5}{16}$ inch wide, ⅜ inch deep, and ⅜ inch between each groove, while ⅜ inch holes are bored through the plate 3 inches from either edge, and every 5 inches lengthways cross-grooves are cut to intersect these holes. On the plates is spread the filter and asbestos cloth, woven a little finer than the ordinary gunny sack. Over this is placed an open wood grating of 1 inch by 1½ inches slats, with openings 3½ inches by 9 inches. This grating and the whole filter are held in place by five heavy brace-pieces 3 inches in thickness, the ends of which slip under the wooden strips bolted to the shell ; small spacing-pieces are placed between the braces, which prevent their coming out or shifting out of place when the barrel is revolving.

All the woodwork of the interior of the barrel is previously boiled in tar and asphalt until saturated, which prevents it absorbing solution and lengthens its life.

Below the filter at each end of the barrel are the valves through which the solution is drawn to the slime-filters or the storage-tanks.

Above the filter, and between the charging holes and the ends of the shell are the valves and connexions through which the wash-water is pumped for leaching.

On one side of the barrel is placed a large storage-tank built of plank and timber, and lined with sheet lead weighing 6 lbs. per square foot, in the bottom of which is an ordinary quartz filter.

Directly below the solution-valves are the slime-filters which are connected with one another by heavy 2 inches acid-proof hose with special connexions. The slime-filters are cast-iron cylinders, flanged on each end, 30 inches in diameter by 18 inches in length, with cast-iron covers, having inlet and outlet-pipes bolted on. The cylinders are lined with sheet lead weighing 8 lbs. per square foot, and have filters similar to those in the barrel, except that above the asbestos cloth a quantity of quartz sand of different degrees of fineness is spread to a depth of 6 inches; and on the top of this there is a second asbestos cloth, arranged to be lifted out and washed when the fine slime and sand have accumulated to an extent which would retard filtration.

In front of the slime-filters are the precipitating-tanks, two for each barrel. They are placed so that the top of the tank is a little below the outlet of the filter. They are 6 feet 6 inches in diameter by 10 feet 6 inches in height, and made of $\frac{3}{8}$ inch tank-steel, with heavy cast-iron flange and cover on the top end, and at the bottom a $\frac{3}{4}$ inch circular plate is flanged and riveted on the outside of the shell, all rivets being countersunk so as to present a smooth surface for the lead lining to rest upon. A 3 inches bolt passes through the centre of the cover and the bottom and a casting placed as a large washer. The cover has a manhole and three 2 inches holes, from two of which lead-pipes extend nearly to the bottom of the tank. The tank is lined throughout with sheet lead weighing 8 lbs. per square foot. There is a 2 inches hole 9 inches from the centre of the bottom-plate, and another 2 inches hole on the side, just above the top edge of the flange of the bottom-plate. The tanks are supported in position on four heavy iron brackets riveted to the shell about 4 feet above the bottom, which rests on a timber-frame, bringing the bottom of the tank about 4 feet 6 inches above the floor.

Between each set of tanks is placed a filter-press with twelve chambers,

19 inches square and ¾ inch distance-frames. The press has a filtering area of 57 square feet. Two heavy lead pipes from each tank, one from the hole in the bottom and the other from the hole in the side, lead to the press, and each pipe has an acid-proof valve close to the tank.

The generators, in which the gas for precipitating is generated, are placed on the floor at the top of the precipitating-tanks. They are plain cast-iron cylinders, of the same size as the slime-filters. The one for generating the sulphurous acid gas is not lined, but has a cast-iron tray and delagrating-plate. The other for generating the sulphuretted hydrogen gas is composed of two cylinders, one above the other, with a plate having a 3 inches hole in the centre between them. The cover of the upper cylinder has a hand-hole 8 inches in diameter in the centre, and a 2 inches hole near one side. The lower cylinder-cover has a 2½ inches hole in the centre, and a small hole tapped for a 1 inch pipe near the top on one side. Both cylinders and covers are lined inside with lead, and a 2½ inches lead pipe is put in the 3 inches hole in the plate that separates the two cylinders. One end is burnt on to the lead lining on each side of the plate, and the other reaches to within 1½ inches of the bottom of the lower cylinder.

The method of operation is as follows:—The ore from the cooling-floor is elevated to the hoppers over the barrels, and from there is charged into the barrel in which the requisite quantity of water and sulphuric acid for the charge have been put. After the ore, mixed with chloride of lime, has been added, the cover is put on and screwed down tight. The friction-gear is engaged, and the barrel allowed to revolve for 1½ to 2 hours, at the end of which time it is stopped, with the charging-holes so placed that the filter is horizontal. The connexions are then made from the pressure-pump to the barrel with the slime-filter and precipitating-tanks. The pump is started and water forced into the barrel, the pressure being seldom above 40 lbs. per square inch. The first water entering absorbs any free chlorine gas left in the barrel after chlorination, and the gold solution is delivered quite clear into the precipitating-tank. Each precipitating-tank has a capacity equal to the solution from two charges. The storage-tank under the barrels is used in the event of there being more solution than will fill the precipitating-tank, and when the filter-cloth wears out and lets the sand through, the solution that accumulates here is afterwards drawn into the precipitating-tank and the gold precipitated.

After leaching the charge in the barrel, it is emptied into a tank below the slime-filter floor, and thence sluiced to the tailings-dump. The asbestos filter-cloth is washed, so as to free it from any sand that may clog

the interstices of the cloth by directing a stream of water under pressure through a small nozzle against every part of it, the water thus put in is discharged by revolving the barrel, which also washes out any tailings that still remain. The life of a filter-cloth is between 50 and 60 charges.

As soon as the precipitating-tank is full of solution, the sulphurous acid gas generator is started and the gas is forced through one of the pipes leading to the bottom of the tank. The gas is generated from sulphur burnt on a tray in the generator, with a current of air forced in at the bottom and deflected over the surface of the burning mass. The excess of air carries the gas through the pipe into the solution, in which it is rapidly absorbed, the air acting as an agitator. When sufficient gas has been passed into the solution to convert all the free chlorine gas into hydrochloric acid, the sulphuretted hydrogen gas is generated from sulphide of iron and dilute sulphuric acid, the acid solution being poured into the lower cylinder of the generator, upon the sulphide of iron placed on a perforated lead plate close to the bottom of the upper cylinder. Air-pressure is turned into the lower cylinder through the hole in the side, driving the acid solution up through the 2½ inches lead pipe into the upper cylinder, where it comes into contact with the sulphide of iron. The air is also allowed to pass through the generator to the bottom of the precipitating-tank, where it acts as an agitator, and collects the precipitated sulphide in a flocculent form. As soon as the precipitation is complete, the air is shut off and allowed to escape from the lower cylinder, when the acid solution recedes from the sulphide of iron, and the gas ceases to be generated. The acid solution is used over again, until saturated with sulphide of iron.

The precipitate in the tank is allowed to stand for about an hour, in which time most of the sulphide of gold has settled to the bottom. The valve on the pipe leading to the filter-press from the side of the tank is then opened, and the supernatant liquor is allowed to pass through the press, in which any sulphide of gold still in suspension is collected. When filtration becomes slow, air-pressure is turned into the tank and the liquor is forced through the press. After four precipitations the precipitate which has collected in the bottom of the tank is forced into the press, through the pipe in the bottom of the tank. To compress the sulphide-cake in the press, air is allowed to blow through till the filtrate stops coming. The press is then opened, the sulphide-cake is removed, dried, roasted, and put away till the clean-up is made, when the accumulation from 15 days' run is put in a crucible with borax, nitre, and a little quartz sand, fused, and cast in a mould.

The amount of precipitants used to precipitate a tank of 2.500 gallons

of solution is—sulphur, 2 lbs.; sulphide of iron, 4 to 5 lbs.; sulphuric acid, 16 lbs.; and 9 gallons of water.

The capacity of the works is 75 tons per day.

Power for the whole of the works is furnished by a 125 horse-power engine and two 75 horse-power tubular boilers, which are largely in excess of the amount required. A small air-compressor and steam pump take steam from the boilers.

The number of men needed to operate the plant is 30, including the chemist and superintendent.

For power, 6 to 7 cords of wood or 4 tons of bituminous coal are required per day; for roasting, 5 to 6 cords of wood are used per day. The chemicals required are—chloride of lime, 8 lbs. per ton chlorinated, and sulphuric acid, 15 lbs. per ton.

The wear and tear of plant, oil, etc., is estimated at between 1s. 1d. and 1s. 3d. per ton of ore treated.

The buildings to cover the plant are—coarse-crusher and dryer, 34 feet by 38 feet; mill, 38 feet by 40 feet; roaster, 40 feet by 62 feet; chlorination-shed, 36 feet by 54 feet; engine and boiler-house, 46 feet by 40 feet. Space is allowed for one more set of rolls, a second roasting cylinder, and two extra barrels, which would nearly double the capacity of the works.

The Golden Reward mill has paid £12,500 in dividends during the year 1892, and is now treating between 90 and 100 tons per day with 4 barrels at work.

Practice in the Otago District, New Zealand.*

The province of Otago, in the Middle Island, was the first gold-milling district opened in this colony. Its history dates from the famous discovery of auriferous gravel in Gabriel's Gully in June, 1861. Though it has ever since continued to be more of an alluvial field than a quartz-mining camp the last few years have witnessed considerable progress in the development of quartz-reefing, the importance of which is becoming better recognized. For the year ending March 31st, 1892, the yield of gold amounted to 105,531 ounces, worth £423,527. The milling practice bears the impress of the system followed in the older centre of Victoria, but altered conditions have induced certain variations which give it undoubted characteristics of its own.

* The writer is indebted for the particulars given of this and the two following districts to papers by Mr. T. A. Rickard on "Variations in the Milling of Gold-ores." *The Engineering and Mining Journal* (New York), vol. lv., pages 78, 101, 222, 247, 389, 416, 534, and 560.

The three mills, statistics of which are given in the following comparative table, are fairly typical. The Phœnix battery at Skippers is one of the best known in New Zealand, and nestles at the base of the snow-capped range of the Southern Alps. The Premier mill at Macetown is an old one, shortly to be replaced by a larger plant. The Reliance is at Xenthorn, a mining township, which, though of comparatively recent origin, has already passed through many vicissitudes of chequered prosperity.

Name of Mill.	Stamps.				Depth of Discharge.	Capacity of Stamps per Day.	Capacity of Mill per Day.	Grating Used.	Holes per Square Inch.	Concentrates.	Bullion.	Retort.	Wear of Gratings.	Loss of Mercury per Ton of Ore Treated.	Consumption of Water per Minute.
	No. of.	Height of Fall.	Weight.	Drops per Minute.											
		Ins.	Lbs.		Ins.	Tons.	Tons.		No.	%	₱1000	%	Days.	Dwts	Gals.
Phœnix ...	30	7½	800	78	3½	1·4	40	Steel cloth	140	Not saved	930	45	6	8·4	4
Premier ...	10	7	750	77	6½	1·2	12		180		949	34	8	7	3½
Reliance	10	7½	850	75	2½	2·0	20		200		850	35	12	5	5

The Phœnix mill, at the head of Shotover river, was one of the pioneers in the utilization of electricity for the transmission of power. The plant was erected in 1884, and consists of two Pelton water-wheels, working under a head of 180 feet, which drive two Brush dynamos, connected by two No. 8 B.W.G. copper wires, forming a line (nearly 3 miles long) to a Victoria motor, which in turn supplies power to run 30 stamps, together with a rock-breaker, and lights the buildings and plant.

Two kinds of grating or screen are used, wire-cloth being chiefly employed, but when the supply runs short the ordinary round punched iron is substituted. The holes in the two cases are of similar size, but the number per square inch is 324 in the one case and only 140 in the other.

The pyrites in the ore has been proved to be of very low grade, and no concentration is therefore attempted.

The loss of mercury was 90 lbs. on a crushing of 3,107 tons of ore. The gold saving is effected by mortar-boxes and blanket-tables, the residues from the one and the washings from the other both undergoing supplementary treatment in an amalgamating-barrel.

The method of milling is extremely simple: the ore passes through a rock-breaker and is fed automatically into the battery, on the principle adopted at Clunes, which has been already described.

The mortar-box has a depth of 9½ inches from the lip to the bottom, divided as follows:—From bottom of grating to top of die 2½ inches;

thickness of die 4 inches, and false bottom 3 inches. This false bottom consists of two sets of four bars, which are placed under the dies, and are packed in between with sand. Each bar is 3 inches square, and has a length of 2 feet 5 inches. The space between the outer bars and the side of the mortar-box is 3½ inches, the width of the coffer being 15½ inches. The arrangement is an expedient for surmounting the difficulties presented by a mortar, whose shape is unsuited to the nature of the milling required by the ore. Before starting the mill the coarse sand from the previous clean-up is packed between and around the dies.

The order of drop of the stamps is 3, 5, 1, 4, 2. No mercury is added to the ore when in the mortar-box, the gold being caught by the action of gravity alone.

On leaving the mortar-box the pulp has 3 drops, making 18 inches in all, before it reaches the uppermost blanket. This fall serves the purpose of spreading the material. There are no amalgamating-tables, and the pulp passes immediately over the blanket-strakes. These last have a length of 18 feet and a width of 6 feet, subdivided into 4 runs. The gradient is ⅞ inch per foot, and about 4 gallons of water is supplied to each stamp per minute, delivered to each set of 15 stamps (3 batteries) by two 1½ inches pipes under a head of 25 feet.

The blankets are freed of the gold and heavy sand that they collect by vigorous rinsing in a tub of water. The top row of the four runs of blankets is washed every hour, and the lower ones are washed at longer intervals, depending on the richness of the ore.

The blanketings or residues from the washing are removed from the tub when a certain quantity has accumulated, and are conveyed in buckets to a barrel (5 feet by 4 feet), having a capacity of 120 gallons (equal to about 1 ton of blanketings). In running 25 to 30 stamps on average 15 dwts. ore, a sufficient supply for a barrel is obtained each day. The water in the barrel is used cold, and 75 lbs. of mercury is added to it. Experiments are being made with sulphate of copper. The barrel turns at a speed of 20 revolutions per minute. When amalgamation is thought to be completed, usually after 24 hours, the material is emptied into a vat; thence it is slowly fed by a running stream of water to a shaking-table of the Rittinger type, 8 feet in length and 2 feet wide. Below the table there are a few pieces of copper plate, but these do not serve much purpose. The collection of amalgam from the contents of the barrel by the shaking-table occupies 2½ to 4 hours, depending on the rate of feeding, and this varies with the heaviness of the pulp.

The amalgam, pyrites, heavy sand, etc., thus roughly concentrated are

placed in enamelled iron buckets to be further washed by hand in a pan. This latter has the shape of the ordinary gold pan, but is made of copper, and its surface having been amalgamated by frequent use, it readily collects amalgam.

Mr. Rickard states that on examining the tailings below the shaking-table he found that they contained a large amount of floured quicksilver. The average loss of 8·4 dwts. of mercury per ton of ore is above the ordinary, and is no doubt due to the high speed of running the barrel.

At the monthly clean-up the battery-residues are roughly screened on a riddle, and the larger bits of quartz removed, previous to adding the residue to the blanketings for treatment in the barrel.

The friable character of the mill-stuff makes the wire gratings used preferable to those of punched iron. It is more easily placed in position on the screen-frame, has a much longer time of wear, and much greater area of discharge. The punched iron gives finer crushing at first, before the burr is worn off, but afterwards becomes easily choked up.

The short life of the gratings, a week for the wire, and slightly less for the punched iron, is not due to anything in the ore itself, which is a comparatively clean quartz, but to the fragments of wood (from mine timbers) which to a quite unusual extent find their way into the mill-stuff. They choke up the gratings which, by the pressure of the water and pulp thus held back, are burst. This is notably the case with the punched iron, which discloses lines of weakness along the vertical divisions made by the press employed in its manufacture. The wire cloth, No. 18 mesh, costs 9s. per yard. It is sold in pieces 30 yards long by 2 yards wide. The grating is cut out of this to the size of 2 feet 6 inches by 10 inches. On the other hand, 120 punched Russian iron gratings cost £17 or 2s. 10d. each, as against 2s. 2d. each for the wire cloth. The expenditure under this head amounts to £5 per month for the 30 stamps.

In place of blankets green baize is used, costing 3s. per yard, and has a time of service varying from three to four months. The expenditure per month is equal to £7 for the entire mill. Those least worn are always placed in the first row. The washing is done by boys who receive 7s. 6d. per shift. One stout lad will do the work demanded by three batteries, but cannot manage the washing of the blankets of four batteries, *i.e.*, 20 stamps.

The ore is essentially of the free-milling type, and is broken from a large quartz-lode traversing schists. The quartz often has a laminated or ribbon structure which enables it it to be readily broken. Inclusions (horses) of country rock are common, and the stone only carries $\frac{1}{2}$ to $\frac{5}{4}$ per cent. of pyrites.

Concentrating tests have shown that the highest grade of concentrates obtained contains only 10 dwts. of gold, giving a value too low for treatment in this locality, and on the small scale required by the circumstances of the case. The gold occurs free. Ore containing more than the usual percentage of pyrites is generally below the average grade,* the gold is not therefore notably associated with the pyrites. It is usually coarse, and often visible. Quartz, which in the stopes does not on examination show occasional specks of gold, is generally of low tenor. On being crushed, the matrix of quartz readily separates from the particles of gold. It might be anticipated that the coarsest gold would be caught in the mortar-box, and that obtained from the first row of blankets would be less fine than that caught on the bottom one, and such is the case. Pieces are found in the battery very coarse indeed, weighing occasionally 5 to 8 dwts.

The mill is lighted by arc lights, and on examining the blanket-strakes by the aid of this powerful light the yellow gleam of the gold particles can be observed scattered over the green baize.

A clean-up shows the distribution of the gold-saving:—230 tons yielded 691 ounces of amalgam, of which 270 ounces came from the blanketings and 421 ounces from the mortar-box residues. On retorting, 313 ounces of bullion resulted, which on melting was reduced to 301 ounces 4 dwts., worth £3 19s. 3d. per ounce.

The most striking features of the milling-practice of the district are that, as a rule, no mercury is used in the mortar, its use being confined to the after treatment, the gold-saving being effected by gravity alone.

It will be allowed that the simpler mill treatment is, the better, because it is usually also cheaper. Another milling axiom is that the treatment should be adapted to the nature of the ore. Here, if the methods employed are elementary, the character of the mill-stuff is no less remarkably simple. Whether the mill succeeds in the extraction of a proper percentage of the value of the gold in the ore is the only question, to be decided. Repeated assays of the tailings from the Phœnix mill prove that excellent work is being done. The composition and character of the ore justify the entire replacement of the ordinary copper plates by blankets, and the successful extraction confirms this method.

In milling, as in mining, one is apt to generalize somewhat hastily, and the good work done by this mill has made the manager an enthusiastic advocate of blankets and an equally pronounced enemy of amalgamating-plates. In an experiment carried out at this mill, two 5 stamp

* In this respect it differs from the ore of many other localities.

batteries were supplied with 80 tons each of the same kind of ore. No. 1 battery was provided with mercury inside the box, with copper amalgamating-tables outside and mercury-wells, and, finally, two rows of blankets; No. 2 battery was supplied with no mercury, and was supplemented by blankets alone. The result of the test showed that 8 ounces (or 2 dwts. per ton) more were obtained by No. 2 than by No. 1 battery.

In condemning copper plates the manager equally objects to the use of mercury in the rest of the mill, and would confine its employment to the final collection of the gold in the blanket-washings. As a case in point, and to confirm the correctness of his ideas, he instanced the Invincible mill on the other side of the same range of mountains where the gold-saving was done by the mercury in the battery itself by wells, amalgamating-tables, and, lastly, by blankets. On ceasing to add mercury to the ore in the mortar-box, it was found that more gold was saved.

The two instances at the Phœnix and Invincible mills merit careful examination. It was scarcely surprising that the addition of mercury to the ore at the Invincible mill, when in the mortar-box, did not improve the gold-saving, nay, that it caused a loss, for the mortar-boxes are merely square iron boxes in no way adapted to the particular work required of them.

The explanation of these abnormal results is to be found in the fact that the mortars were not designed of a shape adapting them to inside amalgamation, and there was no opportunity given to the amalgam to collect out of reach of the falling stamps, but, on the contrary, the quicksilver added was subject to a violent agitation, which caused it to become floured. The particles of mercury thus broken up are readily carried off by the water, and, escaping with the tailings, take with them a certain quantity of gold.

At the Phœnix mill the experiment quoted is vitiated in a similar way, as a mortar-box does not become a successful amalgamating-machine by the mere addition of quicksilver. The batteries of this mill are rectangular in section, with vertical ends and sides, and are in no way adapted to inside amalgamation.

To make a fair comparison between the effectiveness of amalgamation, as against blanket-saving, it is necessary to have the two types of batteries—one roomy and of particular shape, the other narrow and rectangular—kept in view when considering their suitability for the two methods of milling; but there is no suggestion here intended that blankets could be advantageously replaced by copper plates at the Phœnix

mill. Different ores require different modes of treatment, and if blankets will arrest the gold, it is obviously not advisable to use expensive quicksilver or to go to the trouble of copper plates.

The mode of milling at the Phœnix mill is of the utmost simplicity, but it is suited to the ore whose gold contents it is designed to extract.

The Premier mill at Macetown is a much smaller plant, but is engaged in the treatment of a somewhat similar ore by slightly modified methods. The mill possesses 10 heads, weighing 700 lbs. each, and the speed varies from 75 to 80 drops per minute. The height of drop has a maximum of 9 inches and a minimum of 6 inches, according to the hardness of the mill-stuff.

The issue or depth of discharge averages 6½ inches, from 6 inches when the dies are new to 7 inches when worn down. The depth is regulated by the insertion of a blind or blank piece of sheet-iron inside the screen-frame, which increases the issue at the start when fresh dies have been placed in position. As the dies wear down a smaller piece is inserted, and finally the full depth of the screen is utilized. The grating is of round punched iron, 180 holes per square inch. The bullion is 949 fine, and the amalgam yields on retorting 30 to 38 per cent. The gold saving is done by the mortar-box, to which mercury is added, by copper plates on the tables outside, by wells, and finally by blankets supplemented by a berdan pan.

There are no rock-breakers in use, the feeding of the batteries being done by hand. The mortars of the two 5 head batteries are of different patterns. One is more roomy than the other, and therefore discharges the pulp more slowly.

Seeing that amalgamation inside the box is desired, the millman is right in preferring the wider coffer, since it gives more shelter to the particles of gold and mercury. On examination it was found, as might have been expected, that the pulp issuing from the wider mortar was finer than that from the other, although the same kind of screen was used in both.

Contrary to the usual practice, the blankets precede the copper plates; the pulp has a drop of 22 inches to spread it before it falls on to the first row of blankets. These are 12 feet long and 4 feet 3 inches wide, divided by three longitudinal partitions. They slope 1½ inches per foot. The blankets succeed each other in three equal lengths. The first or top row is washed every hour, the second every alternate hour, and the third every third hour. Then follow the copper amalgamating-tables, 9 feet long by 4 feet wide. The total length is subdivided by

five wells, one each at the top and bottom, and three others at equal distances between the plates. Of the five only three are supplied with mercury. They are 8 inches wide and only ½ inch deep.

The residues from the blankets are shovelled from one tub into a second, from which they are fed by a running stream of water into a berdan pan 4 feet in diameter. Instead of the ordinary ball a suspended muller, called the drag, placed at one side of the pan, does the grinding: this modification keeps the grinding and amalgamation separate, thereby preventing unnecessary flouring of the mercury.

A copper plate, 4 feet 8 inches by 2 feet, is placed below the berdan pan with the view of arresting any amalgam escaping in the slimes. At the lower end of the plate there is also a mercury well. The berdan pan makes one revolution for every three drops of a stamp; that is, 25 revolutions per minute, when the average speed of 75 drops per minute is being maintained.

Of the total quantity of amalgam obtained, 60 per cent. is found inside the mortar, 33 per cent. in the blanket-washings, and the copper-tables save the remaining 7 per cent. It was found that by using copper plates below the blankets, instead of a fourth row of blankets, about 5 per cent. more of amalgam was obtained. This is of interest, as proving (what might be inferred) that copper plates are particularly adapted to arresting fine gold, just as blankets are suited to catching it when coarse.

It will be noticed that at none of the mills is there any effort made to concentrate the pyrites. As a rule (the Phœnix ore being a notable exception) the pyrites of the Otago lodes yield a very good grade of concentrates. There is, however, no chlorination or smelting-plant in the province, and any concentrates obtained are shipped to Australia for treatment, at a cost and delay proportionate to the distance. That fact goes far to explain the neglect of this part of the milling.

Both the Phœnix and Premier lodes carry ore, the gold of which is coarse and free; this explains the comparatively crude and very simple method of treatment. Under such favourable conditions blankets are very effectual contrivances for arresting the gold. This system of gold-saving is of very early origin: it was used in America before the discovery of gold in California. The mining districts of the Sierra Nevada borrowed it from the miners of Georgia, and they in turn owed it to those of Verospotak and Nagyag in Hungary. It came back eastward when the discovery of the Gregory diggings started the mining industry of Colorado. It was derived by the millmen of Otago from the mills of Clunes, which, like those of the United States, borrowed it from Europe.

Blankets mark the infancy of milling and belong to the gossan stage of mining; they can only survive those changes in the ore which obtain with increased depth when that ore remains, as rarely happens, unaccompanied by much pyrites, and that pyrites not associated much with the gold.

The Premier mill uses less water than the Phœnix, because the blankets of the latter have less gradient and larger surface. At the Premier mill, mercury is added to the battery-box, while at the Phœnix this is not done. The latter is probably the more correct practice. The gold is coarse and free, and other things being equal, when a large percentage can be arrested by the blankets, it is probable that the still coarser particles which remain inside the battery would be caught there by gravity without using mercury. In both mills the final extraction of the gold from the blanketings is roundabout and clumsy.

In conclusion, while it may appear that the mills of Otago have but little that can be advantageously imitated by those of Colorado or California, because they are adapted to the treatment of an ore of a very simple character, yet the examination of their modes of working may be of value to the millman in causing him to ponder over the why and wherefore of many parts of his own practice, whose advantages he is too ready to accept without always questioning and considering its actual merits.

Practice at Ballarat, Victoria.

In the year 1891, the output of the Ballarat district amounted to 202,704 ounces 1 dwt. 12 grains, of which 127,971 ounces 8 dwts. 9 grains were derived from quartz mining, whilst the remainder was derived from alluvial sources. Dividends amounting to £222,839 15s. were paid during the same period. The quartz yielded on the average at the rate of 8 dwts. 1 grain per ton, while the pyrites (concentrates) contained 2 ounces 3 dwts. 2 grains of gold per ton. The value of the gold ranged from £3 17s. 6d. to £4 3s. per ounce.

The table on the opposite page illustrates the chief features of several of the leading mills.

The Star of the East at Sebastapol is the most productive quartz mine in Victoria, it produced 34,092 ounces of gold, and paid £79,200 in dividends in 1891. The company own two mills, in the newer one the centre stamp drops ½ inch less than the other four, which is said to produce a better splash of the pulp against the grating. The order of drop is 5, 4, 3, 2, 1. When the gold in the ore is found to be unusually

fine the dies are allowed to wear down more than the ordinary maximum of 4 inches, so as to obtain as deep a discharge as possible, the depth at starting being 2 inches.

The gratings carry 200 round punched holes per square inch and last twelve days, but sometimes, when the gold in the ore is very finely divided, gratings with 270 holes per square inch take their place; these wear three to five days.

	Name of Mill.				
	Star of the East (New Mill).	Star of the East (Old Mill).	Britannia United.	New Normanby.	North Cornish.
Stamps—					
Number	60	20	40	20	50
Weight (lbs.)	1,008	784	1,050	784	784
Number of drops per minute	73	75	60	60	72
Height of drop (inches)	8¼	8¼	8	7	8
Average depth of discharge (inches)	3	3	1½	4½	1½
Crushing capacity per stamp (tons)	2	1·5	2·1	2	1·8
Crushing capacity of mill (tons)	120	30	84	40	90
Description of grating	Round punched			Russia iron.	
Fineness of grating, holes per square inch	200	150	120	120	160
Percentage of concentrates (per cent.)	3½	3¼	1	—	2½
Tenor of concentrates per ton	oz. dwts. 3 9	oz. dwts. 3 9	oz. dwts. 1 7	—	oz. dwts. 5 13
Fineness of bullion (per 1,000)	970	970	978	965	935
Retort percentage	46	46	50	70	35
Wear of gratings (days)	10	12	14	12	6
Consumption of mercury per ton of ore (dwts.)	5·7	5·7	2·6	5·6	9·4
Consumption of water per stamp per minute (gallons)	7½	7½	5	5	2½

As a rule the gold is fairly coarse, there being, however, a marked difference in the product of the two lodes worked in the mine. The amalgam from No. 2 shaft, the ore of which averages 13 dwts. per ton, retorts 45 per cent., while the ore from No. 1, which yields 17 dwts. per ton, retorts 48 per cent., the gold being coarser.

In the new mill, the gold is partly caught in the mortar-box by the introduction of a tablespoonful (4 ounces) of mercury every 2 hours. Outside the tables are covered with plain copper plates, and are given a grade of $\frac{7}{8}$ inch per foot. At their lower end there are two drop wells and one shallow well, which catch but little gold when the plates above them are in good order: they simply serve to arrest any mercury escaping from above. Below are blankets. The blanketings—the residues from the regular washing of the blankets—are stacked, lime is added, and they are allowed to stand two days, after which they are re-introduced into the battery with the usual ore-feed. Below the blanket-strakes are eight

ordinary shaking-tables. The pulp, escaping from these, passes over ties or straight sluices. Three berdan pans are used for grinding the skimmings from the wells.

In the old mill, amalgamation is effected in the mortar, and outside on copper plates, which are given an inclination of $\frac{3}{4}$ inch per foot. They are preceded, however, by two, and followed by three wells. Then come two strips of blanket, 8 feet in length, with a gradient of $1\frac{1}{4}$ inches per foot, succeeded by four shaking-tables, one to each battery. Two berdan pans are used for treating the skimmings: the grinding and amalgamation being commenced with a ball and finished with a drag.

The old mill is stated to be doing the better work of the two, the main distinction between it and the new one being that the copper plates of the latter do that part of the gold-saving which, in the old 20 stamp battery, is accomplished by the upper wells. In both cases most of the gold is caught in the mortar-box by gravity, assisted by the free use of mercury; whilst the larger portion of what escapes the battery is caught by the copper plates in the new mill, and in the old plant by the two wells which precede the plated aprons. The plates involve greater first cost and require more attention than the wells, and it is only in exceptional cases where the ore contains a small proportion of sulphides, and the gold is comparatively free, that wells can compete with plates; but where, as in this instance, they are found to do the work required of them, they are to be preferred for the reasons given.

It is the custom to add a bucketful of lime to each 10 heads every two hours, as this is found to prevent the formation of a black scum on the amalgamating-tables due to the presence of base sulphides in the ore.

At the Britannia United mill, located on Bakery Hill, near the spot where the Welcome nugget (weighing 2,159 ounces and sold for £10,500) was found on June 15th, 1858, the ore treated is more free-milling than that of the Star of the East mine. This is proved by the lower percentage of the concentrates, the greater fineness of the gold, and the increased retort yield. The bullion runs exceptionally high, being worth £4 3s. per ounce, equivalent to 978 fine, and owing to the coarseness of the gold the retort percentage averages 50, while it sometimes reaches 65.

The arrangement of the mill is very similar to that of the new mill of the Star of the East mine. One ounce of mercury is added to the mortar-box per ounce of gold in the ore. Immediately outside the battery there is a well, $1\frac{1}{2}$ inches deep and 3 inches wide, containing 10 lbs. of mercury. The copper plates have a gradient of 1 inch per foot. The blanket-strakes are 16 feet long, divided into three; each strip being 17 inches wide, with

a grade of 1½ inches per foot. Quicklime is added to the battery at the rate of 5 lbs. per 5 heads per 24 hours. The battery water is warmed by conducting the condenser water of the engine into the tank which supplies the mill. This practice of using the condenser water in the battery is decidedly bad, for it is certain to carry grease with it.

At the New Chum Consolidated mill at Bendigo, where condenser water is used in the battery, an examination of the launder which carries it outside the mill showed that the bottom and sides were coated with a slimy ooze which must be prejudicial to amalgamation. At the Britannia United mill, using condenser water, the loss of mercury is only 2·7 dwts. per ton of ore, much less than at the Star of the East mill, where condenser water is not employed. These seemingly contradictory results may be explained by the addition of quicklime, which as an alkali is a solvent of grease, and, though not intended as an antidote for the greasy matter in the condenser water, no doubt acts as such. At the Britannia United mill it is added direct to the ore fed to the battery. At the Star of the East mill it is added only, as before stated, to the blanketings previous to their re-introduction into the coffer. It is in general use at Ballarat for the purpose of neutralizing the acidity of the battery water (produced by the partially decomposed sulphides in the stamper-boxes) and keeps the amalgamating-tables in good order, and prevents corrosion of the screens. The desirability of warming the battery water in a climate like that of Ballarat is, however, open to question.

At the alluvial mines of Otago, New Zealand, the use of mercury is hardly known, the explanation given being that mercury will not act in the cold of that region. This is due to the use of hot water in cleaning-up at both mines and mills. The idea is, of course, erroneous, but there is a substratum of truth in it; for amalgamation is usually assisted by heat, and retarded by cold, but only within narrow limits.

At Black Hawk, Colorado, at an elevation of over 8,000 feet, the millmen say that the bitter cold of winter is better for amalgamation on the copper plates than summer heat, because heat thins the amalgam, and the vibration of the mill, caused by the stamps, tends to make the globules of mercury run off and down the surface of the tables. Cold, on the other hand, thickens the amalgam, and tends to keep it in position; nevertheless, in cases of extreme cold, as in Dakota, the water must be, of course, warmed. Hot water has also one beneficial effect in another way, as slimes which will float in cold water will sink in water which is warmed, owing to the expansion of the air-bubbles, that float the fine dust and create slimes.

On the whole, however, while gold amalgamation is benefited in this way by heat, yet below the temperature of boiling water the effects of a small rise are so slight that it is doubtful if the use of warm water be advisable in ordinary gold stamp-milling. It is certainly not to be recommended in a locality with a summer temperature like that of Ballarat, which is often 75 degs. in the shade, when the water in the mill would in fact be giving off vapour.

The writer believes that in tropical climates an advantage might sometimes be found by cooling the water, say, by storing it in underground tanks.

At the New Normanby mill in East Ballarat the ore treated is of a very free milling nature, the gold which occurs in quartz is almost free from pyrites, and is of a somewhat coarse character, as proved by the retort percentage, the yield of bullion being seldom under 55 per cent., and averaging very nearly 70 per cent.

The North Cornish mill is at Daylesford, in the mining district, but not in the town, of Ballarat. The amalgamation is effected by methods similar to those described as in use at the Star of the East mill. Scarcely 11 per cent. of the amalgam obtained comes from the mortar-boxes, most of it being derived from the skimmings of the wells and the blanket sands. The latter are treated in a set of six berdans, the tailings from which go to a Frue vanner. Up to the end of 1890, on a paid-up capital of only £4,000, the North Cornish property had paid dividends amounting to £85,500, but the mill is stated to be miserably incomplete. Although very favourably situated as regards fall, it is unprovided with rock-breakers or automatic feeders, or with a sufficient number of concentrators (Frue vanners), the ore being one that requires very careful concentration. Fourteen vanners are all that are used, whilst 22 of these machines is the least number that should be employed for good work.

The New Star of the East mill and the Britannia United have exceptionally heavy stamps, following a tendency observable in Californian practice some years ago, which has been abandoned in favour of reversion to a lighter pattern. This is an instance of the needless expenditure of time and money in trying experiments which some other district has already carried out. There is no worse waste than the waste of experience in going over the same ground twice.

In the Britannia United and New Normanby mills the gratings are coarser, and the stamps make fewer drops than the others, owing to the gold being coarser and of a somewhat free character. The unusual coarseness of the gold and the absence of pyrites worth concentration

render rougher stamping permissible, and account for the low drop of the New Normanby as compared with the others.

The North Cornish mill uses the finest gratings, since the gold in the ore it treats is the most associated with pyrites.

In the depth of discharge there is the same wide variation to be remarked as in most colonial mills, a factor the importance of which in milling is too little appreciated all the world over.

In screens the colonial mills have been keeping to the follow-my-leader policy, which is the keynote to all that is deficient in their modes of milling. Actual practical tests show that wire-cloth has a much larger area of discharge than punched sheet-iron. The capacity of many of the Ballarat mills would be increased 10 to 20 per cent. by using wire-cloth gratings, and the small extra cost would be more than compensated by the larger duty of the batteries. The full benefit of wire-cloth screens can be best obtained by having a double set of gratings, so that while one is in use, the other can be dried and cleaned with wire brushes.

The closer work done by the Frue vanners (as compared with ordinary percussion-tables) assists in keeping up the grade of the concentrates at the North Cornish mill, which treats the most refractory ore of any of this group of mills.

The fineness of the bullion tells the same story, that of the Ballarat mills being of unusual purity and finer than the Daylesford bullion.

The very shallow but variable depth of discharge of the North Cornish mortar explains the short life of the screens, which only last, on the average, 6 days.

The top of the dies (when put in new) is flush with the bottom of the screen, and the splash is, in consequence, very violent. The dies are allowed to wear down 6 inches. It is probable that the extraction of the gold would be benefited by leaning towards the greater rather than the lesser depth of discharge, and keeping it uniform.

At the Britannia United mill the average depth of issue is no greater, but it varies between 1 and 2 inches. This, coupled with the fact that the gratings are of coarser mesh, explains why they last more than double as long as at the North Cornish mill.

As the New Normanby and Britannia United ore contains a minimum quantity of sulphides, and the ore is crushed coarser, the consumption of mercury is less than that of the other mills.

The quantity of water used varies with the gradient of the amalgamating-tables, which must be regulated by the heaviness of the pulp ; it is also largely dependent on the extent of blanket surface. The North

Cornish mill uses the least water, since the blankets, being followed by Frue vanners, are shorter than those of the other mills using percussion-tables.

At Ballarat the ample capital and adequate ore-supply remove all excuse for the incompleteness of the mills, so far as concerns appliances and arrangement for the automatic handling of the ore. In the feeding of the 50 stamps of the North Cornish mill, 5 men at 30s. per week are employed per shift, representing a cost of £1,125 per annum. For mining companies like the Star of the East and the North Cornish, both owning magnificent mines, paying large and regular dividends, and possessing very considerable ore-reserves, there can be no excuse for the non-employment of rock-breakers, ore-feeding machines, and a proper and adequate concentrating-plant. They stand as monuments of what should be more truly called obstinate ignorance and perverse disregard of modern experience, than dignified by such a misused word as conservatism. It is greatly to be regretted that for reasons all of them illogical and untenable, the mills of such an important mining district should be so out of date and incomplete. In conclusion, therefore, it must be said that while the actual extraction is excellently carried out, the mills of Ballarat are woefully below the ideal, both in the handling of the ore which immediately precedes stamping, and in the after-treatment which succeeds amalgamation.

Practice in the Ovens District, Victoria.

The Ovens district is north-east of Melbourne, near the border-line dividing Victoria from New South Wales. The mines are scattered over one of the most mountainous parts of Australia, and the mills are located beside streams flowing perennially, in pleasing contrast to the dusty dryness which characterizes most of the mining centres of Queensland, New South Wales, and Victoria. The mills are mostly dependent on custom-work, and are unusually small. The accompanying comparative table, given on the opposite page, indicates their chief features.

The Harrietville mill is the most important and largest in the district, and is the only one that does not do custom-work. It is the property of an English company, who own an extensive group of mines, embracing the Tiddle-Dee-Diddle-Dee, Jackass, Mons Meg, etc. The mortar-box is 5 inches deep, and the dies (which are octagonal) are 4 inches thick. When new the depth of discharge varies from $\frac{1}{2}$ to 1 inch, depending on the amount of sand-packing underneath them. On measuring the depth of discharge in several batteries at the close of a month's work, they were

2¼, 3, 1, 1¼, 1, 2¼, and 1¾ inches, or an average of 1¾ inches. The shoes are 9½ inches in diameter and 9 inches long. A shoe weighs 172 lbs., and a die 84 lbs. Both shoes and dies are of best quality white-iron, fagotted, not cast, costing 16s. per cwt. The shoes wear evenly, but the dies irregularly (cupping). Though the ore is comparatively soft, the average wear of iron per ton of ore crushed is 9·8 ounces of the shoe and 3·4 ounces of the die, the wear of the former being excessive. This is to be attributed to the absence of a rock-breaker, and the use of similar material for both shoe and die. The metal of the die should be less hard and more tough than the shoe, an arrangement which would prolong the life of the latter, and promote more even wear of the surface of the die itself; whilst by breaking the mill-stuff to a more uniform size in a stone-breaker, the shoe would be saved much of the violent abrasion that it otherwise undergoes.

	Name of Mill.				
	Harrietville.	Oriental.	Hillsborough	Railway.	Stephens.
Stamps—					
Number	25	16	8	20	6
Weight (lbs.)	700	784	784	720	840
Number of drops per minute	70	55	60	60	50
Height of drop (inches)	8	9	9½	9	9½
Depth of discharge (inches)	2	3	3	4	4½
Duty per head (tons)...	1½	1½	1¾	1⅘	1½
Capacity of mill (tons)	37	20	14	32	9
Description of grating	Round punched			Russia	iron.
Fineness of grating, in holes per square inch	240	220	200	200	250
Percentage of concentrates	1	†	2	†	†
Gold contents of concentrates (ounces) ...	5	†	1½	†	†
Fineness of bullion (per 1,000)	965	940	940	950	945
Retort percentage	36	52	50	45	48
Wear of gratings (days)	18	17	18	20	18
Loss of mercury per ton of ore (dwts.) ...	19*	8	4	8	8
Consumption of water per stamp per minute (gallons)	5	4	3½	4	4½

For breaking ore in the first instance, a jaw-crusher must necessarily be better adapted to the purpose than a falling weight like a stamp (fine-crushing is not in question), and by compelling the latter to do work which should have been previously done by the former, the material of shoes, dies, screens, and other wearing parts is wasted, the crushing capacity of the mill is diminished, and amalgamation is checked.

The coffer (mortar-box) is of peculiar design, and is provided with an end-discharge. It is 4 feet 6 inches long and 1 foot wide, the interior

* Including pans. † Not saved.

being protected by a cast-iron lining 1 inch thick in four pieces. The bottom of the mortar is flush with the table outside, and it is provided in front with a removable section for cleaning it out.

The grating-frame is in three sections, the two end sections being curved, but each representing a discharge-area corresponding with the one in front, which is 2 feet long. The end gratings have 175 holes per square inch, and the front ones 240. They wear rather longer than those at the end. A set lasts from three weeks to a month, putting through 125 to 200 tons of ore. Complaints in this and other mills are made of the weakening of the gratings along perpendicular lines, caused by the press used in their manufacture. The gratings are not fixed to a frame, but are held in by the latter, which is made of a flat strip of iron bent to the shape of the opening in the mortar-box, and fastened to it by claw-clamps attached to strengthening-ribs, which separate the middle from the side-openings.

The two ends of the mortar-box make a curve, which is an arc, struck from a point in the back of the box. This arrangement of the mortar admits of employing in front, copper tables of unusual width, and by so doing increases the area of amalgamating-surface and the thin distribution of the pulp. An amalgamating table 6 feet wide and 12 feet long is a sight to gladden a millman's heart. The wash of the pulp over the tables is very regular in speed, and even in distribution. The end discharge is generally condemned, and for this reason : that it is ordinarily attempted under the unfavourable conditions of a mortar-box of rectangular shape; the result being that the issue from the end gratings is weak and irregular, whilst the discharge from the front is injuriously affected. These difficulties are overcome at Harrietville by the variation in shape of the mortar-box, and by using screens of larger mesh at the ends than in the front, as the splash against the former is less than in the front of the box.

The gold-saving is done by collection in the mortar-box, by outside tables and mercury-wells, and indirectly by a modified variety of the Rittinger percussion-table: the concentrates collected being first roasted and then ground in pans. Though there are no amalgamating-plates inside the coffer or mortar, mercury is fed into the battery in variable quantities, depending on the richness of the mill stuff. The average amount is about 1 ounce per battery of 5 stamps every half hour.

The amalgamating-tables have a slope of $\frac{7}{8}$ inch per foot, and are subdivided into four sections lengthways. The first, or apron-plate, is 22 inches long, and the one next below is 3 feet long. These are lined with silver-plated copper plates, whilst those below are usually covered with

plain copper plates. A variation in this arrangement, however, enables one to make an interesting comparison of the effectiveness of plain *versus* silver-coated plates.

In front of No. 5 battery the succession is (1) silver-plated copper plates; (2) plain copper plates; and (3) plain copper plates. The middle plate of this set came from the old Mons Meg mill, and was thoroughly amalgamated and in first-class order. In front of No. 4 battery (next to No. 5) the usual order of two silvered copper plates, followed by two plain copper plates, obtained. After a year's work on ore of uniform character it was found that the plain copper plate forming the third plate of No. 5 battery was well amalgamated, while the corresponding copper plate in front of No. 4 was scarcely whitened by amalgam, proving that the second copper plate of No. 5 had not arrested gold and amalgam as successfully as the corresponding silver-plated copper plate, at No. 4 battery.

Following the first copper plate there is a drop-well, 3½ inches deep, succeeded by a shallow ripple, whilst each of the copper plates below is followed by a ripple of similar kind. All are charged with mercury, and cleaned up once a week. Each battery requires 3 bottles of mercury; or 15 bottles is the stock needed by the entire mill. The loss of mercury is about 75 lbs. avoirdupois per 1,100 tons of ore, or 19 dwts. per ton, including the pan-amalgamation of the roasted concentrates. The consumption in actual milling would be about 8 dwts. per ton.

There are five shaking-tables running at 135 strokes per minute, one to each battery.

The ore yields about 1 per cent. of concentrates, chiefly iron pyrites, and the concentrates run 5 ounces of gold per ton, or at the rate of 1 dwt. per ton of crude ore.

The general clean-up on the last day of each month takes 6 hours; the sand collected in the battery being washed over the amalgamating-tables, and thence passed over the shaking-tables. The heaviest portion is returned to the battery on re-starting.

Thirty-three per cent. of the amalgam obtained is collected from the boxes, and of the remaining two-thirds, 8 per cent. comes from the wells, and 58 per cent. from the amalgamating-tables. Of what is saved by the plates fully 90 per cent. is caught in the first length. Of the total yield of gold 86 per cent. is extracted by direct amalgamation, and 7 to 12 per cent. from the concentrates.

The retort yield of bullion varies from 25 to 52 per cent.

The end discharge and clayey nature of the ore explain the large consumption of water.

The framework of the mill consists of light iron standards, and owing to the care taken in building the foundations, the vibration is not excessive. The tappets are keyed on, as it has been proved that the old-fashioned screw-tappet, though very excellent in theory, and answering admirably when first put in, as soon as it becomes at all loose, (which eventually is certain to happen,) is soon worn out and ruined, necessitating frequent stoppages to cut fresh threads.

The mill, with the exception of being unprovided with a rock-breaker, is an excellent one and excellently managed.

The concentrates are roasted in an ordinary reverberatory furnace, with a hearth 27 feet long and 9 feet wide. It is subdivided in its length into three divisions, with a drop between each of 2 inches. The charge is 12 cwts., taking 8 hours to roast, and the daily capacity of the furnace is 4 tons.

The amalgamation-plant, consisting of two Wheeler pans and a settler, is just below and in front of the roaster, and its extraction is exceedingly good, ranging from 90 to 97 per cent. of the gold in the concentrates, as shown by assay.

It seems probable that if the Harrietville mill were equipped with rock-breakers and self-feeders, and a front-discharge mortar with screens of 175 mesh, it would crush more than 1½ tons per 24 hours, with 700 lbs. stamps given an 8 inches drop and 70 drops per minute, or at any rate with slightly heavier stamps falling more rapidly from a less height.

The extreme thinness of the mortar-bottom and excessive weight of the stamp-shoe (172 lbs.) in a 700 lbs. stamp are to be remarked. In America an 850 lbs. stamp usually has a shoe of 125 lbs. The Australian proportion therefore gives an extremely light stem (2½ to 2¾ inches) or tappet, details in which American practice has been controlled entirely by results of experience.

The reason for using finer screens in front than at the sides of the Harrietville mortar is the desire to equalize the volume of discharge. The intention of the end discharge is to cause an even distribution of the pulp over the amalgamating-tables, whose excessive width is thus rendered serviceable. The suggestion has been made, that a copper plate 6 by 8 feet might be better than one 4 by 12 feet, and it seems very sensible. The copper plate should be as wide as, if not slighly wider than, the screen discharge.

As a rule the mortar-opening exceeds 4 feet. If the amalgamating-table is much wider than the discharge, it will be found hard to obtain an even distribution of pulp. The other extreme is, however, more common. Too often plates are less wide than the discharge. In one mill they were

2¾ inches less than the latter, causing a decided eddy along the sides of the table, owing to the pulp being confined to a narrower space than it was discharged from. The increased force of the current thereby caused was certainly prejudicial to the arresting of the gold. In all milling operations the sooner the gold is caught the better, therefore a table which is wide, but over which the pulp is evenly distributed, and which consequently arrests the gold early in its career over the amalgamating-surface, is to be preferred to one less wide but longer, whose gold-saving is continual for a greater distance from the battery. An amalgamating-table is rarely too long; it is poor economy to have it even a little too short. An old copper plate can generally be sold for more than its first cost, and therefore any supposed economy in this direction should be disregarded.

The Combination Process.

The plant and process of treatment which obtains at the re-habilitated mill of the Standard Consolidated Mining Company, of Bodie, California, contains many points which are of great interest, as it differs in some respects from the ordinary, and exhibits another phase of Californian practice, affording us an illustration of the combination process.

The Standard mill contains 20 stamps, and appears to have cost £10,800, or £540 per stamp. Sundry additions were made to the original machinery by the erection of two boilers, 16 feet long and 54 inches in diameter, with brickwork chimney. The mill was originally provided with only one set, necessitating a stoppage of the entire mill for 18 to 24 hours every two months for repairs, which entailed a proportionate loss of product, while expenses ran on as usual. The cost of putting in these boilers was £120, whereas a stoppage of the mill for 20 hours involved a direct loss of £80 to £120; consequently this provision should more than pay for itself in four months. The plant was also increased in 1891 by fitting up two pans and a settler (for the treatment of concentrates), and the addition to the continuous system of two pans and another pointed settling-box.

The ore is crushed wet, and run over silvered copper plates, which, according to the records of the six months prior to the issue of the company's report on January 31st, 1892, took out 80 per cent. of all the free gold and silver contents. The major part of the remainder stays in the batteries, being too coarse and heavy to be thrown out through the screens, and is extracted by amalgamation in the clean-up pan at the end of every month.

The pulp leaving the plates is run over four Frue vanners with belts 6

feet wide, and the tailings from these are led into pointed boxes in order to get rid of the surplus water from the batteries and vanners before treatment in the pans. The thickened pulp discharged from the settling-boxes is raised by means of a bucket elevator to a series of eight pans and three settlers arranged on the Boss continuous system. From the settlers, after passing through an agitator, the tailings escape into the tailing-pits.

The summary of mill work for the year 1891 shows 15,704 tons of ore crushed. The average crushing during the first five months was 1,249 tons, during which time a No. 10 (corresponding to 40 mesh) Russian iron slot-screen was used on the batteries. The average crushing for the last seven months was 1,351 tons, or a gain of 102 tons per month accomplished by the use of a No. 0 (corresponding to 40 mesh) tin punched screen. These latter cost 2s. 10d. each delivered, and two of them will outlast the average Russian iron screen, which costs 13s. 8d.

While battery samples are taken regularly at the mortar-lip, they serve simply as an indication of the grade of the ore, but do not represent its true value, on account of the heavier particles of gold remaining in the mortar, as already explained.

The total value of the ore is obtained by adding the bullion product of copper plates and batteries, the amount of monthly output of concentrates (calculated from daily assays of weighed amounts) and the value in the vanner tailings, this latter item being the product of the number of tons of ore crushed, multiplied by the average tailings assay for the month.

While the percentage of extraction is not high, it is, according to the few available records, from 4 to 5 per cent. greater than that obtained on this ore previous to the present system of milling, and exceeds that of 1890 by 3 per cent.

Analysis shows the ore to contain no base metals other than a very small amount of oxide of iron; nevertheless, it is only partially free-milling, and the percentage of extraction is consequently low.

Richer ore, such as that milled in the spring and summer months, gives higher results on account of carrying a higher proportion of the precious metals in a free state. While, treating a lower-grade ore, as during the last half-year of 1891, the extraction could only be maintained by the partially-successful amalgamation of tailings in the continuous pans. Working on a lower-grade ore a higher extraction was obtained than in previous years, which is certainly satisfactory considering that the ore-treatment is here limited to a certain line of milling operations by the conditions which obtain of high labour, expensive fuel, and costly freights.

The following table shows the expenses for 1891 :—

Month.	Cost per Ton of Ore.			
	Mining.	Milling.	Superintendence, Bullion Freight, Insurance, Taxes, Office Expenses, etc.	Total.
	Dollars.	Dollars.	Dollars.	Dollars.
1891. February ...	6·208	2·982	0·475	*9·665
March	7·665	3·742	0·366	11·773
April	7·784	3·973	0·334	12·191
May	7·840	3·830	0·524	12·194
June	8·149	4·930	0·854	†13·933
July	6·005	4·361	0·876	11·242
August	7·042	3·388	0·564	10·994
September ...	6·727	3·719	0·527	10·973
October	7·770	4·570	0·610	‡12·950
November ...	6·904	3·903	§1·418	‡12·225
December ...	6·842	4·013	0·554	11·309
1892. January ...	6·295	3·370	0·675	10·340
Averages for the year {	7·080 29s. 6d.	3·890 16s. 2½d.	0·650 2s. 8½d.	11·620 48s. 5d.

The accompanying statement, showing the cost per ton of mining, milling, and all incidental expenses, exemplifies the expensive conditions alluded to. That milling is being done at a much reduced cost is shown by the fact that 15,704 tons of ore were crushed in 1891, at a total milling expenditure of 61,002·94 dollars, or 3·89 dollars per ton, while in 1889 only 11,498 tons had been crushed at a cost of 60,000 dollars, or over 5·20 dollars per ton.

AMALGAMATION OF 86 TONS OF CONCENTRATES.

86 Tons.	Assay Value per Ton.			Contents based on Assay of each Pan Charge.			Percentage of Extraction	Loss of Mercury per Ton.
	Gold.	Silver.	Total.	Gold.	Silver.	Total.		
	Dols.	Dols.	Dols.	Dollars.	Dollars.	Dollars.	By Assay. Per Cent.	Lbs.
Concentrates ...	67·53	28·18	95·71	5,807·28	2,423·49	8,230·77	74 $\frac{8}{10}$	2
Tailings ...	15·34	8·77	24·11	1,318·99	754·70	2,073·69	Actual.	
Assay difference	52·18	19·41	71·59	4,488·29	1,668·79	6,157·08	78 $\frac{8}{10}$	
Actual bullion returns {	Actually extracted per ton of ore.							
	55·62	19·11	74·73	4,783·56	1,643·39	6,426·95	78 $\frac{8}{10}$	
	Actually contained in tailings per ton of ore.							
	11·91	9·07	20·98					

* Cost exceptionally low, in consequence of not including the entire month's expenses, the bullion product being unusually small.

† Cost unusually high, due to payment of all outstanding bills.

‡ Increased, on account of laying in winter supplies for mine and mill.

§ This item higher due to payment of taxes.

The four concentrators produce 10½ tons of concentrates per month, carrying 80 to 100 dollars per ton, *i.e.*, £18 15s. on the average per ton. This represents a contents of concentratable material in the original ore of ⅝ per cent., showing the ore to be hardly of a concentratable character. Nevertheless, the vanners cost so little to operate while furnishing a high-grade product that it pays to continue running them.

There were on hand in July, 86 tons of concentrates, the product of eight months' work. Previously these concentrates had been shipped to the Selby Silver and Lead Company of San Francisco, and on account of heavy freight and reduction costs had yielded to the Standard Company only £9, or 48 per cent. of their value (£18 15s.), this arrangement, therefore, appears to have cost the company £9 15s. per ton.

It being found that the concentrates contained no sulphides and little or no base metal, two pans were fitted up in the mill, and the concentrates were put through a specially adapted slow treatment, with the results shown in the previous statement, and a return of 31 per cent. more than that obtained by shipping to reduction-works.

The tailings from the pans are carefully banked up separately in a reservoir, and will yield a further proportion of their contents after exposure and oxidation.

A Carter magnetic separator, through which the dried concentrates were being passed in order to eliminate the magnetic oxide of iron (of which they contain 5 to 10 per cent.), was discarded, as pan-amalgamation tests showed 16 per cent. higher extraction on the original concentrates than those which had gone through the separator, though the bullion produced was necessarily baser.

The concentrates are now being regularly treated by pan-amalgamation as the best and cheapest method of realizing them quickly.

Chlorination-tests, after roasting with salt and sulphur, have shown no higher percentage of extraction than that of slow amalgamation; moreover, the loss of gold and silver in roasting is considerable (15 to 20 per cent.), and the cost of wood at 10 dollars per cord makes the process more expensive than amalgamation, for which the ore is fitted.

The mercury on the plates and in the pans takes out all the free gold that will amalgamate, the remainder needs a different treatment, and is worked as tailings. For this purpose the mill originally contained six pans and three settlers fitted up for the Boss continuous system, and one pointed settling-box 11 feet by 7 feet by 7¼ feet in size. One or two trial runs had been made previous to 1891, but the product was increased by mixing in some richer material from the blanket-sluices in the Bulwer Standard mill.

The pan capacity being too limited, two pans were added to the above plant in 1891, and another settling-box put in, to take the overflow from the first. Though there is still a loss of slimes in the overflow of the second box, it is impossible to avoid this and retain the proper consistency of pulp in the pans.

It may be here noted that the mill produces an abnormal quantity of slimes, on account of the large quantity of clay accompanying the quartz in the veins. From trial runs of several weeks each it was ascertained that basing the bullion by the use of salt and bluestone in quantity did not increase the extraction, and the needful small amount of chemicals necessary was determined. The tests also showed that much grinding in the pans was to be avoided. Since then the work on the tailings has steadily improved. The average of several chlorination-tests on the raw tailings show about 8 per cent. higher extraction than the continuous-pan system, and for the months of October, November, and December, 1891, and January, 1892, gave a yield of 35 per cent., at a cost of 3s. 8d. per ton.

Most probably, roasting with salt and sulphur, and chlorination by either the Plattner or barrel process, would give a much higher percentage of extraction, but with wood at 10 dollars per cord, labourers' wages at 3·50 dollars per day, and freight rates of 3 and 4 cents. per lb. (the latter has to be taken into consideration on account of chemicals essential to either process), it would be impossible to figure a profit on tailings averaging 7·50 dollars in gold and silver per ton. Large sample lots of tailings were sent to the Denver Gold and Silver Extraction Company (cyanide process) for trial, and to the Noble M. and M. Company, of New York,* in the hope of obtaining a method giving a higher extraction, without roasting or other great expense. The cyanide method after 12 hours' agitation in a ½ of 1 per cent. solution gave only 46 per cent. extraction, which is just 12 per cent. better than the present cheap pan process, a margin that would be more than swallowed up by the greater cost of the process. On 100 lbs. of concentrates sent for trial also, worth 86 dollars per ton, a test of 12 hours' agitation in a 1 per cent. solution gave 88 per cent. extraction, or 9 per cent. in excess of that obtained by pan treatment, a difference which would not cover the cost of the cyanide.

During the year ending January 31st, 1892, 16,336 tons of ore was mined, including 8,658 tons of filling from the old stopes. The total cost of working per ton was:—Mining, 6·14 dollars; ore transport, 0·113

* A 200 lbs. sample of vanner-tailings when treated, gave only an extraction of 38 per cent.

dollar ; milling, 3·525 dollars ; superintendence, etc., 0·804 dollar ; a total of 10·582 dollars. The mill ran 357½ days, crushing 16,336 tons of ore, the average duty of the stamps being 2·42 tons per day. The stamps weigh 750 lbs., have a drop of 6 to 7 inches, and a normal speed of 104 drops per minute. The average value of the ore was 17·93 dollars in gold and 1·99 in silver, or 19·92 in all. The amount saved was 72·8 per cent. Of the free gold 85·5 per cent. was obtained from the apron-plates, and 14·5 per cent. from the battery-sands. There was 123½ tons of concentrates treated, having an average value of 58·97 dollars in gold, and 30·86 dollars in silver, a total of 89·83 dollars per ton. These concentrates yielded by pan-amalgamation 80·6 per cent. The tailings left to weather from the previous year were ploughed over several times, and then amalgamated, 134 tons being put through.

A very base bullion was purposely made, barely 100 fine in gold and silver, but the extraction was very good, averaging 67 per cent., and raising the total extraction on the original concentrates to 93 per cent. The yield of the mill tailings treated in the Boss continuous pans was 67·2 per cent. of their assay value, or an average of 12·47 dollars per ton. The use of acid, in place of salt and bluestone which have proved efficacious, was discontinued as it was found more expensive, and gave no better results.

(To be continued.)

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THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.*

By A. G. CHARLETON.

Part VII.—Medium-coarse to Fine-crushing.

Stamps.

Dr. R. W. Raymond, than whom no one perhaps has contributed more to the scientific literature of practical mining, makes the following remarks with regard to stamps† :—

"In considering the economical application of stamping-machinery, we meet at the beginning with serious difficulties in obtaining accurate information for comparison. The weight and fall of stamps vary as the shoes and dies wear out, and this alone may lead to a change of speed.

"Moreover, defects in engines, boilers, and machinery, for the transmission of power, may occasion serious loss, which cannot fairly be charged to the arrangements of the stamps proper.

"Finally, the hardness and tenacity of the rock crushed varies so much that comparisons between different localities cannot be implicitly trusted. The safest experiments are those made in the same mill by changing first one and then another condition of working, but this is seldom possible for such conditions as weight and lift of stamps, and only within narrow limits for their speed. We may eliminate questions of friction, transmission, and generation of power, in the case of stamps, by measuring the power actually developed by their fall. Thus the weight multiplied into the fall in feet, and the number of drops per minute, gives us exactly the number of foot-pounds exerted by each stamp. Dividing by 33,000 (the number of foot-pounds per minute per horse-power) we have the horse-power per stamp, from which the effective power of the whole mill may be obtained. Dividing the amount of rock crushed daily by the effective horse-power gives us the daily amount per horse-power, and this is the best measure that can be obtained for the effectiveness of stamps.

* *Trans. Fed. Inst.*, vol. iv., pages 233 and 351; vol. v., page 271; vol. vi., pages 69, 295 and 457.

† *Mineral Resources West of the Rocky Mountains*, 1871.

A complete discussion of the subject would require us to determine the exact influence of the discharge, etc., and the exact resistance offered by different classes of rock, for both of which points the data are wanting."

Prof. Munroe, summing up his paper on the weight, fall, and speed of stamps,* says the following conclusions may be drawn from the data at hand:—

"1. The product of a stamp-battery, other things being equal, is directly proportional to the foot-pounds developed in the fall of the stamp, and to the number of blows per minute, provided only that the discharging capacity of the battery be equal to or greater than its crushing capacity.

"2. Both the force of each blow and the number of blows per minute may apparently be increased per minute up to certain limits with but slight diminution of the rock crushed per horse-power, provided that the discharge be increased in the same proportion.

"3. The discharge of a battery depends on the character of the screen (its area, the size and number of its openings, etc.), and on the number and duration of the splashes produced by the fall of the stamp.† The discharge cannot therefore be increased indefinitely, but up to a certain limit increases in proportion to the speed of the battery, or with the number of drops per minute.

"4. With a limited discharge, equal to or less than the crushing capacity of the battery, the product may be raised by increasing the speed, but cannot be raised by increasing the force of the blow.

"5. A light stamp with a high drop and a heavy stamp with a low drop produce nearly equal crushing effect, provided that the foot-pounds developed are the same. The advantage, however, is in favour of a heavy stamp and a low drop.

"6. To increase the crushing effect of a stamp-battery it is better to increase the weight of the stamps than the height of the drops.

"7. A drop of 8 inches has been generally adopted as giving better results than higher drops. It is possible that still lower drops may prove economical.

* *Trans. Am. Inst. Min. Eng.*, vol. ix., page 84.

† To this writer the author would add—the slope of the screen-plate, ratio of total area of stamp-shoes to total area of mortar, height of screens above the dies, and slope of screen-opening.

"8. The objection to high drops does not seem to apply to high velocity of impact as obtained by spring, atmospheric, or steam stamps, which apparently give results as good as or even better than those from stamps of which the foot-pounds are due principally to weight.

"9. High speed and high velocity of impact lessen the weight of the stamps, and diminish the number of batteries requisite for a given amount of work, and therefore lessen the first cost of the plant. These smaller batteries do not demand large buildings for their accommodation, nor the construction of large and expensive foundations. To offset this saving, must be placed the extra wear and tear due to high speeds.

"10. In the ordinary construction of stamp-batteries with cams and tappets, velocity of impact can be secured only by giving the stamps a high lift: this is incompatible with high speed, but, as we have already shown, under the conditions of limited discharge, speed is more to be desired than high velocity of impact.*

"11. With steam-stamps, atmospheric-stamps, or spring-stamps, both high velocity of impact and high speed can be secured. By the use of machines of this character the maximum effect can be obtained with the smallest plant. The product that can be obtained from one head will be limited by mechanical conditions, and by the impossibility of increasing the discharge beyond a certain limit. The last limitation will operate sooner with fine-screens than with coarse.

"12. In fine stamping both the difficulty of discharge and the work of crushing increase with the fineness of the product. According to Mr. Rittinger the product of the stamp-battery is proportional to $\sqrt[5]{d^3}$, in which d is the diameter of the screen openings. This ratio seems to hold good with the stamp mills of this country [America], though data are wanting to determine this exactly."

The subject has also been ably discussed by Mr. A. N. Rogers, in a paper on "The Mines and Mills of Gilpin County, Colorado."† He

* This has been confirmed by experience at the Caledonia mill, Dakota. It does not hold good, however, with dry stamp batteries where the blow of the stamp both pulverizes the ore and helps to force it out of the mortar. At the Metacom mill, for instance, it was found that with 60 drops per minute, not quite 1 ton per stamp was discharged per 24 hours, 90 drops discharged 2 tons, and 102 drops a little over 3 tons per stamp, the increase in speed thus increasing the yield by 244 per cent.

† *Trans. Am. Inst. Min. Eng.*, vol. xi., page 29.

takes a somewhat opposite view to Prof. Munroe (who supports the Californian system), and argues, from mill practice in Colorado, that light stamps of, say, 500 lbs. dropping 16 inches 80 times per minute are superior in effect to 700 lbs. stamps dropping 10 inches 65 times per minute ;* and he brings forward several practical reasons for his opinion, amongst others that with a low drop and high speed there is great liability for the sands to pack at one end of the mortar, leaving the dies bare at the other, as the agitation in the box gives the ore no time to settle, resulting in loss of work, injury to the mill, and defeat of amalgamation.

In examining the two sides of the question it must be recollected that the duty or amount of stone crushed by any battery, in a given time, depends on the number and weight of the stamps, their height of fall, and number of drops per minute, to which must be added their order of drop, the design (sectional area and position, as well as height of screen-opening) of the mortar, the mesh of the screens, whether hand or mechanical feeding is employed, the quantity of water used, and the regularity or otherwise with which the shoes, dies, and screens wear.

The effect of the more rapid drop in California is to give a less intermittent action, in which it most nearly approaches the idea of a good crushing machine.

It is most important in this connexion that the shoes and dies should wear down flat and evenly ; that the height of drop should be kept constant, and that the bottom of the screen-opening should be regulated to correspond with the wear of the dies, which is effected in Dakota and elsewhere by using a chuck-block. A maximum discharge appears to be obtained by dropping the middle stamp first.

In Colorado, the depth of water, in the deep form of mortar used, exercises an important influence in deadening the blow of the stamp, and weakening the force of the splash, giving the gold time to settle, retaining the pulp in the box, and discharging it in a much finer condition than is the case with a less deep discharge using the same size of mesh screen.

The author's view of the case is, that the choice between light stamps

* Practice has been somewhat modified since Mr. Rogers wrote, both in Colorado and California. In the former locality the rate of drop still remains the same, but the weight of the stamps has been increased to 550 or 600 lbs., and the drop to 18 or 20 inches. In California, on the other hand, the drop has been diminished to between 4 and 6 inches, increasing the weight of the stamps to 750 or 850 lbs., and running them faster, at from 90 to 105 drops per minute. In some cases (as at the Grass Valley mill) the drop runs up to 7 inches, and the stamps weigh occasionally as high as 900 to 1,000 lbs. with lower drop.

with high drop and low speed, and heavy stamps with low drop and high speed can only be settled by knowing the character of the ore, whether it can best be crushed fine or coarse, and deciding on the nature of its treatment according as a large output or close saving of free-gold is the vital point to be considered.

The Colorado practice has been developed and proved correct, using a deep roomy mortar for fine-crushing, and amalgamating mostly in the battery; while the Californian practice, which aims at crushing coarse and catching a large proportion of the gold outside the boxes has equally strong, if not stronger, features to recommend it (with certain grades of ore) on the Pacific slope.

The question, in fact, to be determined first of all, is whether the stamps themselves are to be used as a close-saving amalgamating-machine, as well as for crushing, or whether the latter object is the one to be kept most in view—trusting to extraneous gold-saving appliances, to catch whatever gold escapes the stamp-mortar.

Whilst at the present time the theoretical work done by California and Colorado stamps is about the same, for the reasons that have been stated the actual work done does not correspond in the two cases. In Colorado the stamp crushes 1 ton per 24 hours, while in California with stone of similar hardness the amount varies from two-and-a-half to three times as much.

The relative wear and tear of iron is shown by Mr. Rogers (from examples cited) to be greatest in the Californian mills, amounting to 1·8 lbs. worn from the shoes per ton of rock crushed, and 0·9 lb. worn from the dies, compared with 1·44 lbs. loss of shoe-metal, and 0·31 lb. of die-metal in Colorado; assuming the ore reduced in both cases to the same standard of fineness.

As the Californian battery-pulp is much the coarser, in reality, the figures in actual practice are however reversed, 70·8 tons being crushed in Colorado per shoe, and 78 tons per die, before removal, whilst 79 tons per shoe, and 100 tons per die, are crushed in California. The proportionate breakages are 1 stem per 25,000 tons stamped in Colorado, and 1 stem per 864 tons in California.

Mr. Rogers argues, from the foregoing data, that the destruction of metal per unit of work will be inversely as the efficiency of the battery, regarding the energy stored up in the stamp, as a force, which, if not expended on the rock, will occasion an abnormal wear of the surrounding metal. On this assumption, from various figures given, he deduces: first, that the Grass Valley mill (taken as an example of Californian practice)

wears out most metal per ton of rock crushed, and has the greatest proportion of wear on the die; second, that the Bobtail (Colorado) mill performs one and one-half times as much work for each pound of metal worn from the die, and twenty-three times the work for the breakage of a stem as the other mill; third, that the shoe of the Bobtail mill wears four and six-tenths times as fast as the die, thus proving that the blow is taken by the rock, and not passed through the stone to the die beneath; fourth, that while the Bobtail mill wears the shoe four and six-tenths times as fast as the die, the other mill wears its shoe but twice as fast as the die, which indicates that more of the work passes through the rock into the die, employing the same as an anvil; fifth, that the relative endurance of the stems in the two cases must be taken as the most conclusive evidence that the blow of the Bobtail stamp, notwithstanding the velocity of the impact due to greater drop, has been absorbed in the rock, and the stem has not received the violent shock, which would result from falling upon the metal of the die; and sixth, that by careful analysis of these data no undue wear can be detected in the battery or foundations of a high drop mill as compared with a low drop one, and therefore its work must have been properly expended upon the rock in the battery.

Theoretically, these arguments are interesting and plausible, but it strikes the author, that to give them practical weight, for relative comparison between the two different systems, we ought to know the relative hardness of the rock taken for the test, the relative quality of the metal used for shoes, dies and stems, and the construction of the battery-foundations in each case. The relative endurance of the stems alluded to by Mr. Rogers in his fifth contention, may perhaps be accounted for, by the different taper or diameter of the end (just above which, the stem generally breaks) or to want of attention in annealing the stem after welding.*

To express a fair opinion, it is also necessary to know the position of the guides, and the relative attention paid to them, as if they are allowed to wear or become loose, undue breakages will result. Moreover, the not less important question of feed must be considered, for if the stone banks-up in one end of the box, leaving the dies bare at the other, or is overfed, it will tend to a similar result, though the author admits that there is less probability of this happening with a high drop, than with a low one. In other words, less attention is required in feeding in the former case.

* To change a stem, it saves much time and trouble if a differential pulley be used, attached to an overhead crane running on rails the whole length of the batteries, at a sufficient height above them.

The slow drop and turn of the Colorado stamp makes the shoes and dies wear less evenly than in California stamps; the interval between each drop in the former case being 2 seconds, whilst in Californian batteries it varies from $\frac{3}{5}$ to $\frac{2}{3}$ second.

Regarding the Colorado and Californian systems, without any prejudice either way, there is another point which the author thinks should not be overlooked, viz., the previous and subsequent treatment of the pyritic portion of the ore which contains gold.

Where a certain portion of the gold-ores of a district are of a sufficiently massive character (as they are in Colorado) to be capable of hand-selection, it frees the battery of a certain amount of work, and on this account capacity is not so much an object as it is when every particle of payable stone requires to be stamped. The portion of the ore which thus escapes the battery treatment contains the bulk of the heavier pyrites, and its removal therefore indirectly influences the results of concentration, and the after disposal of the concentrates upon which the manipulation of the ore directly depends. With fine-stamping for instance, such as obtains in Colorado, owing to the free-gold present being very fine and intimately associated with the pyrites, one would expect to find the losses in pyritically-combined gold, greater in concentrating them on end-bump table or blankets and buddles, than with the coarser stamped Californian ores, for which mechanical belt-concentrators are almost exclusively used.

These losses are neutralized, however, in Colorado practice, by the removal of the heavier portions of the ore (containing much of the pyrites) before it is subjected to the stamp-process, and the Colorado pyrites being generally less valuable than the Californian the losses (chiefly of silver) in concentration are less felt and apparent. Further, while the hand-picked pyrites and fine concentrates are well adapted to the smelting treatment,* which obtains in Colorado; they† would be far less, if at all fitted, for the chlorination method, which holds its own on the Pacific slope, as the selected ore would have to be all crushed dry with stamps, or rolls, and the fine concentrates would tend to clog the filters.

* The low tenor of the concentrates in gold, and the proportion of silver they contain, coupled with a preponderance of copper pyrites and grey copper, and the presence of arsenic in the ore, with abundant fluxes close at hand, are conditions which render a matte-smelting far better adapted to these ores than any other, whilst the chlorination process is most suitable, on the other hand, for the siliceous Californian concentrates, in which iron pyrites predominates.

† The author does not of course here refer to ores (carrying considerable quantities of lead), which would be quite unsuited to chlorination, but only to pyritic ores, comparable with those of California.

The gangue of the Gilpin county mines in Colorado is more feldspathic than quartzose, being the product of the alteration of the granitoidal-gneiss and quartz-andesite dykes which penetrates it.

The mill-stuff treated in Amador. Calaveras, and Tolumne counties (California) on the other hand, is of a different description, essentially quartzose, carrying 1 to 2 per cent. of pyrites, but mixed with a large percentage of country-rock, which in this case is slate, augite-schist and diabase, the slate predominating, and the gold being coarser and less intimately associated with the pyrites.

It may be noticed, that the proportion of smelting to free-milling ore, raised in Gilpin County, Colorado, during the year ending May 31st, 1880, was as 10,218 to 113,427 tons, the selected stone having an average value of £10 1s. 6d., as compared with £2 1s. 3½d., that of the mill-rock.

According to Prof. Egleston,* the distinguishing feature of the auriferous smelting-ores of Colorado, is the presence of manganese,† which, while it is oxide on the surface, produces below the water-level beautiful crystals of carbonate, and also by the presence of copper (carrying both silver and gold) instead of lead,‡ as is usually the case elsewhere. The relative fineness of crushing in the Colorado and Grass Valley mills, is

* *Metallurgy of Silver, Gold, and Mercury in the United States*, page 406.

† Manganese often forms part of the gangue of gold and silver veins, and when not superabundant may be considered a favourable association for the precious metals.

‡ By far the greater portion of the silver ores of Colorado are reduced by silver-lead smelting in water-jacket furnaces. This is due to the metallurgical skill shown in decreasing the loss in smelting, by lowering the percentage of lead required, and by the unusual economy with which the process is carried on. Of late there has been a larger use of calcining-furnaces for the preliminary treatment of refractory ores to be smelted. The ores are not reduced at the mines, but are shipped either as crude mine ore or are concentrated at the point of reduction, and the concentrates sent to customs smelting works. Even when the ores are dry (that is containing at most a low percentage of lead) they are usually sent to the smelters for admixture with other ores, with better metallurgical and commercial success than if treated by amalgamation or lixiviation. This practice is largely due to the usual failure of attempts to operate silver-lead furnaces on the ores of single mines or districts, and to the practice of selling ores and concentrates through sampling works which partly conduct a commission and brokerage business by correctly ascertaining the value of the consignments, and selling the lots to the highest bidder. This system has been specially advantageous to the smaller mines, as it obviates the cost of erecting reduction works and affords immediate cash returns for the producer. The smelting plants established at central points, such as Denver, Leadville, Durango, and Pueblo, are thus enabled to obtain as nearly as possible a typical smelting mixture at all times, which can be worked to advantage, and large quantities of ore can be treated which could not be profitably smelted alone. The chief difficulty with this system is the cost of freight when the ore has to be brought from outlying districts, but this diminishes as the country is opened by roads, and railways.

stated to have been as 80 to 40 mesh, only 1·17 per cent. of the Bobtail ore being caught on a 40 mesh-screen, while the fineness of the tailings of the two mills was as 3 to 2.*

Mr. Rogers says, that the saving by amalgamation above the blankets was fully 70 per cent. of the contained value of the gold in the ore, and about 6 per cent. of the silver, and the cost of milling but little more than 4s. 2d. per ton,† embracing current expenses, repairs, and renewals.

The average expense in Colorado, in the early days, was stated (the writer believes, by Dr. Raymond) as :—

	£	s.	d.
Crushing and amalgamating mill-rock	0	16	0
Treatment in pans and dolly-tub ...	0	0	7
Concentrating and hauling	0	0	11
Extracting gold from concentrates ...	0	3	0
Total per ton	1	0	6

The summary of operations for 1878,‡ shows that 22,936 tons were crushed at the Bobtail mill, out of which 75·80 per cent. of the gold was saved by direct amalgamation, and 58·96 of the mineral was caught in the concentrates. The loss of mercury has been stated by some authorities as being largest in the Colorado practice, but Mr. Rogers combats this idea, alleging that in well-managed batteries, it only amounts to 0·02 pound per ton of ore.

The width and roominess of the Colorado mortar present conditions favourable for allowing the pulverized ore to escape the falling stamps, and assisted by the deep discharge, retains the finely crushed pyrites inside, whilst the long drop gives the interval of time necessary for the gold to settle on the plates in the box where it is amalgamated.

Mr. Rickard considers the results of an interchange of treatment using Californian batteries on Colorado ore, or *vice versa*, would be as follows :— The Gilpin county ore is worth, say 8 dwts. or 8 dollars per ton. The local methods extract 5·60 dollars by amalgamation, without counting subsequent concentration, at a cost of 70 cents. A Californian mill crushing over 2½ tons of ore, at a cost of 25 cents per ton in California, would give an extraction in Colorado of only 4·00 dollars, but would crush the Colorado ore three times as fast as a Colorado battery, so the cost would be, say 25 cents for a net yield of 3·75 dollars as against 4·90 dollars obtained by present methods. Using the Gilpin

* According to Mr. T. A. Rickard, it has been proved by actual test, using a 40 mesh-screen, that fully 70 per cent. of the pulp in Colorado and 50 per cent. in California will pass a 100 mesh-sieve.

† Table II. shows that the cost at the present day is still less in some mills, as for instance the Hidden Treasure, where it only costs 3s. 3d. per ton.

‡ Statement by Mr. G. W. Gray.

county mill in California, treating an ore containing 6 dwts. and worth 6·00 dollars per ton, the Californian mill would extract 70 per cent. at a cost of 35 cents, leaving a balance of 3·95 dollars per ton. The Colorado mill would extract an increased percentage, say 75 per cent., but the ore being harder it would crush less, and hence the cost per ton would be greater than in Colorado, say 1·00 dollars per ton, leaving a net yield of 3·50 dollars per ton. The Californian mill would, if crushing 100 tons a day, therefore show a profit of 45 dollars per day more than the Colorado battery. Other factors that have influenced the choice of methods have been the smaller size of the Colorado ore-bodies as compared with those of California, whilst the construction of a Colorado mill of a capacity equal to a Californian plant, would require double the capital outlay.

Outside of Colorado and Californian methods, there is a third system of milling. The author refers to the Dakota practice* of employing 850 lbs. stamps, dropping 9 to 12 inches, running at 74 to 85 blows per minute, and using a mortar-box, which possesses peculiar features of its own, whilst it meets, to a certain extent, the Colorado requirement of depth. The principle to be observed is to feed low; the feeders seeing that the height of the ore, between the shoes and the dies does not exceed an inch, and as much less as possible, without the shoes commencing to pound. The screens used are 30 mesh wire, or corresponding slot-punched.

The loss of iron from wear of shoes is 0·35 to 0·37 lbs., and from dies 0·40 to 0·48 lbs. per ton, using chilled white-iron shoes, and grey mottled iron (chilled at the top) for the dies. The loss of quicksilver is usually 0·0011 to 0·0044 lb. per ton of ore crushed, and not more than 5 stems are required yearly for a 60 stamp-battery. A careful record of all wearing parts of the machinery about a mill and their individual life, ought to be kept by the management, if operations are to be conducted in a business-like manner.

With close-fitting wooden tops to the boxes, catch-pans, under-bearings, and care in the use of grease in places where it might drop into the mortar, there ought not to be much trouble from that source. The cams, under ordinary circumstances, should be lubricated with axle-grease, and a curtain spread below them, to catch any grease that they may throw off whilst in motion.

The groves in the guides are lubricated with a preparation of black-lead and linseed oil, mixed warm, in such proportions as to form a stiff

* With an ore containing any considerable quantity of pyrites (the same as in California) care would have to be taken not to overcrowd the concentrators in adopting this method.

paste. Oak guides last eighteen months, and pine only two. Failing the use of Broughall or Fargo sectional-guides, instead of grooving the ordinary guide-blocks to receive the stem, a good practice is to cut a dove-tailed recess opposite each stamp, in which a wooden concave-faced key can be fitted, so that the grain of the wood is parallel with the stem instead of across it.

In wet crushing 1 to 1¼ tons of ore is an average amount to crush per horse-power. The amount of fuel required to run the stamps, depending on their weight and the load of the other machinery, averages 0·22 to 0·25 cord of wood per stamp.

A novel feature in the Homestake practice has been the introduction of screens of aluminium-bronze, which are stated to have proved very satisfactory as a substitute for Russian-iron. The bronze, supplied by the Cowles Electric Smelting and Aluminium Company contains 5 per cent. of aluminium, 95 per cent. of copper, and a trace of silicon, and is furnished in unperforated sheets at 1s. 10½d. per lb. The sheet is 0·038 inch thick, lasts six months, and does not break or get unserviceable through wear, while the life of an ordinary Russian-iron screen seldom exceeds two weeks, and is sometimes much less. This may be an item of economy, small as it may seem, worth noticing.

The choice of any special make of automatic-feeder, it has been shown, chiefly rests on whether the ores are dry, or damp and sticky, as some classes of feeder suitable for the one, are not so for the other.

As the mills of Dakota have to treat low grade ores, it is necessary for their profitable operation that large quantities should be dealt with as rapidly as possible, saving as much gold as practicable by the simplest means. Consequently, the mills in the Black Hills amalgamate both inside and outside the mortar in order to avoid the more expensive auxiliary operations necessary to recover the gold used in Californian mills. It is evident that the Dakota practice differs from that of both Colorado and California in that it permits of the large capacity of the Californian mill, and at the same time of adopting the simple plan of amalgamating inside the box to advantage (as is done in Colorado), hence it is a sort of compromise between the two systems. The discharge of the Californian mortar being so shallow (only about 4 inches), inside plates are not admissible, the constant and violent agitation scouring the amalgam off them and preventing the gold settling.

The proposition of employing light stamps with a small area of shoe to mitigate dead-stamping, is a suggestion the author does not agree with, and, in fact, it has been shown that the proposed cure is calculated to

aggravate the disease. To escape dead-stamping, the point is to disentangle the particles of gold, as far as possible at one blow, without the head requiring to drop a second time, and to at once get the pulp out of the mortar-box. This is most likely to happen with a head of heavy weight and large area, providing the battery openings are made of sufficient size, and screens of proper mesh are employed, for facilitating the ready escape of the ore. With these precautions, a low discharge, and a sufficient supply of water to clear the boxes without flooding the plates, dead-stamping need never happen, if the stamps are speeded right, and dropped in proper order, with sufficient lift; at any rate, what little there is, ought to be regarded as a lesser evil, than reducing the duty of the battery by diminishing the weight and area of the head. Dead-stamping is, of course, more liable to occur if the gold in the ore be coarse.

Dead-stamping is least liable to happen with ore like that of Clunes, where the gold occurs in seams and cavities in more or less honeycombed quartz. With such an ore Mr. Rickard points out that a certain blow will break the brittle quartz and liberate the ductile gold. But such ideal conditions are rare, and more frequently the stamp has to deal with material of such uneven size as regards the metal and the gangue that the particles are apt to become confused together, and the one is often crushed too much and the other too little. Owing to the greater friability of the sulphide minerals, the latter are generally reduced to much greater fineness than the stone, and hence if the gold is not very intimately associated with the pyrites, or in too fine a condition, coarse stamping is often sufficient to liberate it.

On the Continent, where rectangular non-revolving heads* are employed, the tendency of late years, at Clausthal for instance, has been to set the heads as close together as possible, to give the greatest possible crushing-surface between shoes and dies. The Caledonia mill in Dakota, using stamps 9 inches in diameter, weighing 850 lbs., and dropping 85 times per minute, claims to save 85 per cent. of the free-gold in the stone (60 per cent. of which is caught on the inside plates); and that there is but little dead-stamping, is vouched for by the fact that the batteries put through no less than 4½ tons of quartz, mixed with mica and amphibole-schist, per head

* The rotatory action of the stamp (making with high-speeded batteries a whole revolution in 9 to 10 drops, and with slow stamps one complete turn for each fall), whilst it may tend to abrade the surface of the gold and rub off any film of foreign matter likely to interfere with amalgamation, produces the bad effect of tending to slime any sulphides present, which accounts for the preference in Germany for non-revolving heads; stamps being used there mostly as an auxiliary crushing-machine, for after-concentration of the ore.

per diem. Mr. Melville Atwood has drawn attention in an article, entitled "The Microscope in Metallurgy,"* to the deleterious effect which even a small quantity of zinc (blende), if present in the ore, produces in battery-amalgamation through sickening the mercury. He also states that when a large proportion of pyrites is present in the ore of the Bodie mine he found that the battery-water had a decidedly acid reaction, due to the presence of soluble ferric and ferrous salts. The zinc destroys the action of the quicksilver, enfilming it with a compound insoluble in water, preventing the metallic contact so necessary to amalgamation taking place between the detached mercurial globules. The same happens if arsenical pyrites be present.

In experimenting on some of the Wynaad ores, the author found great difficulty in keeping the copper-plates free from yellow-stain (until thoroughly coated with gold-amalgam), and he attributes that circumstance to the softness of the water, causing it to absorb a large amount of air, which even more, perhaps, than carbon dioxide tends to form ferric sulphate (the most active agent in staining the plates) when in contact with the pyrites. As a remedy in similar cases, a small quantity of caustic lime might be introduced into the water-supply tanks. The water referred to had a hardness of 2 degs. by the Clark test, and contained 6·5 grains per gallon of solid matter consisting of carbonate and sulphate of calcium and a trace of silica. Air acts prejudicially in another way, as it is naturally absorbed in the water by the violent action of the stamps in the form of bubbles; and whilst the quartz is broken into irregular grains, owing to the more highly-developed cleavage of the metallic sulphides, they are broken up into thin grains and flakes, which the air has a tendency to carry off in suspension. Artificially raising the temperature of the water, and so causing the air to expand and the bubbles to burst, is undoubtedly the only means of overcoming this difficulty.

In connexion with copper-plates, the outside aprons ought certainly not to be cleaned oftener than absolutely necessary. The gold-amalgam should, in fact, be allowed to settle on them, forming ridges which present surfaces for other particles of gold and amalgam in suspension in the water to catch upon and cling to. The amalgam must not be allowed, however, to become too hard, or pieces of it are liable to break off and be swept away. Should this happen, it must be at once rectified by throwing a thimbleful or two of mercury inside the battery, or sprinkling a little over the copper-plate. Nearly every millman has his own ideas about the

* *California Mining and Scientific Press*, June 24th, 1882.

arrangement, preparation, and care of copper-plates, and such like matters. Loss is liable to occur by giving the tables too much or too slight a gradient,* and by using an excess or an insufficiency of water, both of which points must be carefully attended to. In some mills, as elsewhere stated, silvered copper-plates and rocking-tables are used (with a riffle attached) for catching truant amalgam. It is a good plan to put a perforated pipe over the ordinary riffles. directing a number of water-jets on to them to prevent clogging, if, as sometimes happens, the turning on of more water into the box itself would make the pulp too thin, preventing proper settlement of the gold, and discharging the ore in too coarse a condition. So long as there is no actual movement of the tables longitudinally or laterally, the general vibration caused by the rise and fall of the stamps produces a kind of pulsation in the water flowing over them, assists gravity, and is favourable to the settlement of the gold and amalgam. The advantages of stamps, to the writer's mind, consist in their simplicity of mechanism, durability, and large crushing capacity, coupled with the fact that if one or more heads get out of order it does not throw the whole mill out of gear; nothing being more fatal to economy than a whole plant standing still, since every minute lost, multiplied into the total staff employed, soon mounts up to a large figure in pounds, shillings, and pence.

Then, again, the repairs are light, and can be mostly done by a good blacksmith ; the capacity of the mill can be regulated to constant variations in the supply of ore by starting or hanging-up a battery (5 heads); and, finally, stamps extract a large percentage of gold in those cases where it is free, without any accessory machinery.

The preliminary step towards banking gold, is like jugging a hare, to first catch it with as little delay as possible, and that is what is done in the mortar of the stamp-battery. The man who christened the South African gold formation *banket*, perhaps took a somewhat sanguine view of the case, if he expected to do without the intervention of even the simple stamp, as a means to that desired end ; but he evidently had the right principle in his mind's eye.

So long as 70 to 85 per cent. of the free-gold contents of the stone can be extracted with stamps, independent of all other appliances but copper-plates inside and outside the boxes, and a few mercury-traps, the battery (a good old conservative piece of mechanism) will remain a formidable

* The heavier the ore the greater the gradient required. The inclination most effective for settling the gold may, however, cause the pyrites to settle, and too heavy a gradient may sweep both away.

rival to many so-called improved processes, for which a liberal royalty has not unfrequently to be paid, for the doubtful privilege of experimenting with them. Its simplicity and capacity explains the extreme cheapness of the process in such instances as the Homestake group of mines, the Alaska Treadwell, and the Morgan mine in Wales.

For the figures the writer has been able to give in table II. part IV. of this paper,* in regard to the last-named property, he is indebted to Prof. C. Le Neve Foster, Mr. N. T. Williams, and the makers of the machinery, and he is pleased to be able to state, that Wales, as represented by the Morgan mine, under Mr. Williams' management, can claim to have crushed gold-quartz more cheaply than any stamp-mill in the world.

The expense of producing gold at the mines of the Golden Leaf Co., at Empire in Montana, may also be specially cited as an instance of cheap mining and milling. The cost of mining and milling per ton is given by Mr. H. M. Beadle as follows:—

	Empire. 1891.	Empire. 1892.	Bell Boy. 1892.
	s. d.	s. d.	s. d.
Mining	4 2	4 2½ ...	7 10½
Mine development	2 0	1 5 ...	1 5½
Milling	3 3½	3 3½ ...	3 2
General expenses (haulage included)	0 10½	1 10½ ...	5 3½
	10 4½	10 9½	17 9½
Quantity mined and milled ... Tons	53,700	46,600	10,880

For the sake of comparison, the cost of crushing in some of the best known California stamp-mills is given below:—

Name of Mill.	No. of Stamps.	Cost per Ton. s. d.
Moore	10 ...	3 4
Stickles...	20 ...	1 8
Sheep Ranch	30 ...	4 2
Idaho	35 ...	10 0
Bunker's Hill	40 ...	2 6
Keystone	40 ...	3 1½
Plumas Eureka	60 ...	2 0
Plymouth Consolidated ...	160 ...	1 7½

In the three instances specially alluded to, amalgamation is carried on in the battery, and no extraneous appliances are used, but copper-plates, riffles, and mercury-traps.

Stamps certainly do possess some defects however. They are not suited for instance for an ore (*a*) which contains pyrites in large quantity, which cannot be sorted-out beforehand, (*b*) which requires to be pulverized very coarse, (*c*) or which requires for its after-treatment to be gradually reduced in several successive stages to a certain size; for such work, rolls have a decided superiority. For direct and continuous

* *Trans. Fed. Inst.*, vol. vi., page 76.

operation, grading the material closely, pulverizing finely, evenly, and cheaply, discharging the waste as soon as disengaged (thus freeing the amalgamation of a hindrance, and the mill of an encumbrance), as a well-known authority has remarked, stamps have, however, never yet been matched, whilst they take advantage of the brittleness of the rock in crushing it.

Mills.

The Huntington, Schranz, and Tustin mills, and edge runners, like the improved Chilian mill, as well as Frankfort mills, have some advantages over stamps, as their tendency is to rub and brighten the gold, which renders it more easily attacked by quicksilver.

Considering the results it is reported to have achieved, the Schranz mill seems to deserve to be better known than it is, outside of Germany. Its essential parts are a large slightly coned ring or annular plate, revolving about a vertical axis at the rate of 12½ revolutions per minute, and three conical rollers, with fixed inclined axles, which are radial with respect to the central vertical axis of the machine. The rollers are set on the plate 120 degs. apart, and rotate by frictional contact. An illustration of the machine is given in Mr. Linkenbach's *Aufbereitung*. It claims to minimize the production of fine meal and pulp, requires 3½ horse-power to run it, and working on 0·12 to 0·32 inch (3 to 8 millimetres) quartzose jig-tailings, has been proved to have a capacity of 3,200 lbs. per hour, with a consumption of 31 gallons of water per minute (5¼ gallons out of this amount being used for the sizing-drum). The durability of the wearing parts (the crushing-surfaces being of the toughest Bessemer steel) is a special feature. For ore not exceeding ⅓ inch in size, it seems as if this mill ought to take the place of stamps in some concentrating works, as experiments in reducing ore of 5 and 8 millimetres (0·197 and 0·315 inch) size down to 2·4 millimetres (0·01 inch) both by stamps and by mill, gave the following interesting results:—

Proportional Quantities of Ore of Different Sizes produced in Crushing, expressed in Percentages.

Millimetres Inch ..	3·2-2·4 0·126-0·094	2·4-1·6 0·094-0·063	1·6-0·9 0·063-0·035	0·9-0·5 0·035-0·020	0·5-0·2 0·020-0·008	Less than 0·2 Less than 0·008	Second Pulp Settlings.	Finest Basin Settlings.
Stamps...	—	4·68	15·15	16·96	24·08	16·72	3·88	12·66
Schranz mill ..	6·95	21·07	26·27	16·92	15·31	7·21	1·71	1·83

This table shows that in using the Schranz mill, 64·26 per cent. of the material was obtained in a settling run, between sizes 2·4 and 0·5 millimetres (0·094 and 0·020 inch); 15·31 per cent. consisted of fine

meal and coarse pulp, and 12·98 per cent. was fine pulp; while the corresponding figures for coarse stamping were 36·78, 24·08, and 39·13 per cent.* The writer may also observe that Frankfort and Chilian mills are both an exceedingly good form of combined comminutor, agitator and amalgamator, more so, he thinks, than pans for gold-ores, and in proof of it, he need only point to the good work the former are doing in Italy (mentioned by Prof. C. Le Neve Foster) and his own observation of the latter in Mysore. In fact, if gold sulphides must be ground and amalgamated raw, he considers a Chilian mill one of the best machines to employ for that purpose.

The chief case in which the Huntington mill distances other competitors is in the treatment of comparatively soft clayey ores, which neither a stamp-batttery nor rolls are well adapted to handle; whilst conversely, of course, the Huntington mill is not as well fitted as stamps for crushing very hard rock.

The writer thinks that the stamp-battery is *par excellence* the machine for most brittle ores, just as rolls, and, in exceptional cases steam-stamps, are for tough ones, but he would give the preference, as he has remarked, to the Huntington mill for soft clayey ores, especially such as contain rusty-gold. The portability of this crusher, the comparative cheapness with which it can be set up and used, just like a stamp battery, without mixing, grinding, or other auxiliary plant, or elaborate superstructure, must specially recommend it in those cases where it is desired to give a free-milling ore a fair trial in an out-of-the-way place, at a minimum expense. In evidence of its economy the author may cite the record of the Spanish mine, California, mentioned by Prof. Egleston,† as it is the cheapest instance of gold-ore reduction on record, costing per ton :—

	s.	d.
Labour of tending and feeding, cleaning-up, and superintendence	0	4·50
Water for mills and rock-breaker	0	1·80
Shoes	0	1·45
Screens	0	0·65
Dies	0	0·85
Caps, scrapers, and bolts	0	0·10
Renewals of working parts	0	1·00
Mercury, at £8 6s. 8d. per bottle	0	0·25
Oil (illuminating and lubricating) for mills and breaker	0	0·10
Depreciation	0	0·50
Cost of milling per ton	0	11·20
Labour at breaker	0	1·00
Wear and tear at breaker	0	0·25
Total cost per ton	1	0·45

* Mr. Kunhardt "Ore-dressing in Europe," *School of Mines Quarterly*, series 2 (New York).

† *Metallurgy of Silver, Gold, and Mercury, in the United States*, vol. ii., page 419.

The gangue of the Spanish mine is a decomposed slate, carrying stringers of quartz, with clay-partings between the slate and the quartz, and the ore is very poor, only averaging about 2s. 6d. to the ton.

During the last four months of 1886, the four mills employed at these works crushed 136 tons per day. The mill is driven by an 8 feet Pelton wheel, with 15 inches of water and a pressure of 320 feet, at 60 revolutions per minute; 22 horse-power being required to run the plant, which consists of a Blake crusher, Challenge feeders, and three 5 feet and one 4 feet Huntington mill. Seven shoes a month are worn out by the four mills. The screens are No. 5 slot, and last only four days. The 5 feet mills grind 35 to 40 tons a day each,* and the smaller one 30 tons. Mercury is added to the mills in the proportion of 1 ounce to 4 tons of ore, and 1 ounce is lost for every 15 to 30 tons of ore crushed, the exact amount depending on its richness.

The entire plant cost as follows :—

	£	s.	d.
Four Huntington mills and self-feeders	1,322	2	7
Labour in erecting and building	480	3	5½
Silver-plated amalgamating-plates	413	12	3½
Waterpipe and wheel-shafting and pulleys ...	254	6	5½
Lumber of building and flume	248	17	2
Hardware...	213	15	4
Blake crusher	128	16	10½
Cost of milling-plant under cover and running	3,061	14	2

Freight cost £5 a ton from San Francisco, lumber cost 9s. 4½d. per 100 feet.

The total cost of mining and milling, estimated on a production and reduction of 4,047 tons in 29 to 30 working days, during November, 1887, was 2s. 2d., which gave a profit of 6½d. per ton (on mining and milling) at the Spanish mine: a result which probably eclipses any that has ever been obtained from a low-grade gold-ore. 45 per cent. of the gold caught is saved round and inside the mills, and 55 per cent. on the copper-plates outside; very little gold amalgam it is said is flowered.

Mr. T. B. Morse expresses the opinion with reference to Tustin mills, which he states have worked satisfactorily at the Willard Mine, Calaveras county, California (where they have been used for crushing ore for amalgamation on copper-plates, and concentration of the sulphides), that a comparison of the two methods of crushing with stamps and mills, shows that a much larger proportion of slimes are produced by the former. As a consequence, he found in trials made with a very hard ore, in which the gold was exceedingly fine, and the sulphides soft and brittle, that a much greater percentage of fine gold was amalgamated after the mills than after

* This seems above the average, which is usually reckoned at 20 tons with a 5 feet mill.

the stamps. He put the saving in concentration after stamping at only 18 to 20 per cent. of the assay value of the ore, whilst with mills he claims to have saved 85 per cent. With the same ore and with the same screen he considered that one mill was equivalent to 6 or 8 stamps.*

GRANULATORS.

Direct crushing in one process has been effected by two very ingenious machines, the Sturtevant mill, and the Gates granulator. The former consists of a chamber of cast-iron fitted with perforated screen plates having oblong openings about ¼ inch wide, fitted on the top with a hopper for feeding ore. From opposite sides two cups or cones of chilled iron enter the chamber through openings provided with exchangeable collars; and the cones revolve in opposite directions at about 900 revolutions per minute. The ore, which must be absolutely dry, is charged in lumps of probably about 4 cubic inches; and when the chamber is filled the ore packs itself against the revolving cones, and the grinding action commences. The ore is rapidly crushed and passes out through the slots in the screen-blocks. The dust is drawn off by an exhaust fan connected with the hopper under the machine. The product is screened and the returns go back to the mill to be reground. The capacity of the machine with a suitable material that does not require to be crushed too fine is enormous. A mill with 20 inches cups or cones will (it is stated) crush about 20 tons per hour of calcined ore to 12 mesh, at a cost of 11d. per ton. The Wear amounts to only about ¼d. per ton crushed, but on hard ore the wear is very considerable, and rolls in such cases are found more economical. The mill is said to require about 49 to 115 horse-power, according to the hardness of the ore, on the other hand it simplifies the plant and reduces the first outlay. The Gates granulator is the same in principle as the Gates crusher, but the segments composing the outside hollow cone are supported by springs yielding under a certain pressure. Samples of ore crushed by this machine show a very superior and even granulation.

Mr. Sahlin and Mr. S. R. Krom† give a number of interesting figures by which the work of the Sturtevant mill as compared with crushers and rolls may be gauged. Mr. Sahlin, experimenting with Port Henry iron-ore crushed by the Sturtevant mill, passed it over the identical sizing-screens used for the ore granulated by crushers and rolls, with the following results:—

* *Report of the California State Bureau*, 1886.

† *Trans. Am. Inst. Min. Eng.*, vol. xxi., page 534.

Size of Particles. Mesh.	Crushers and Rolls. Pounds.	Per Cent.	Sturtevant Mill. Pounds.	Per Cent.
10–16 ...	310	31·0	22½	22·50
16–24 ...	210	21·0	21¼	21·25
24–40 ...	79	7·9	9¼	9·25
40–60 ...	175	17·5	29½	29·50
Above 60	209	20·9	15½	15·50
Loss ...	17	1·7	2	2·00
Totals ...	1,000	100	100	100·00

This test appears to decide in favour of crushers and rolls, but it is not a fair one, the conditions being altogether favourable to the Sturtevant mill. Mr. Sahlin states that the fine dust had previously been all drawn off by the exhaust fan in the Port Henry sample. The ore as received was absolutely clean and free from dust, whilst it is said that the rapid accumulation of worthless dust in the collecting room is a serious source of loss and embarrassment. Moréover, the Port Henry ore crushed in the Sturtevant mill was a clean, clearly defined, coarse crystalline magnetite, containing only a small percentage of dust-producing minerals, while the heterogeneous mixture treated in the above experiment by the rolls did not carry over 30 to 38 per cent. of iron, and contained besides a large amount of dust-producing materials, such as felspar, olivine, and apatite, all of which remained in the sample when sized. Another point in favour of the crushers and rolls was the smaller amount of power required for a certain hourly production. To illustrate this, a 20 inches Sturtevant mill (stated to have a capacity of 20 tons per hour), with requisite auxiliary machinery, is compared in the tables below with a crusher and roll plant of similar capacity. It is assumed that both plants reduce the ore to 10 mesh size.

CRUSHER AND ROLL PLANT.

	Horse-power.
Coarse-crusher (24 inches by 14 inches) ...	20
Two fine-crushers (5 inches by 20 inches)	20
Three pairs of rolls	45
Elevators and conveyors	10
Loss in transmission (30 per cent.)	29
Total	124

STURTEVANT MILL.

	Horse-power.
Coarse-crusher (20 inches by 14 inches) ...	20
Sturtevant mill (20 inches)	110
One pair of rolls	15
Elevators and conveyors	10
Loss in transmission (30 per cent.)	46
Total	201

This shows a saving of 41 per cent. in favour of crushers and rolls; and in a day's run of 20 hours, 77 horse-power saved, represents a saving of 6·93 dollars or about 1¾ cents per ton of ore crushed, calculating 3 lbs. of coal per horse-power per hour and coal at 3 dollars per ton.

As to repairs, it is more difficult to find a basis of comparison between the two systems. At the Port Henry mill, it is said that the wear on the cups or cones amounted to one new set per 600 tons of ore ground. These cups are said to weigh about 800 lbs., and their cost can certainly not be less than 3 cents per lb., or 24 dollars per set, which is equal to 4 cents per ton for grinding-cups alone, irrespective of cost of screen-blocks, collars, bearings, etc. The life on hard ore of a pair of 26 inches steel tyres, costing about 200 dollars, averages, as near as can be ascertained, from one to two years. Using three pairs of rolls in the above assumed 20 ton plant, which will actually run 20 hours per day during 250 days, crushing yearly 100,000 tons of crude ore, and estimating the average life of the tyres at one and a half years, one set of new tyres would, at this rate, be required for every 50,000 tons of ore crushed. The cost of tyres in this case will therefore amount to 0·4 cents per ton.

The following tests were made by Mr. Krom for granulation and dust. A sample of Port Henry ore ground in a Sturtevant mill to pass a 10 mesh screen ($\frac{1}{10}$ inch diameter hole) gave the following results:—

	Per Cent.
From 10 to 100 mesh (granular)...	86¼
Through 100 mesh (dust)...	13¾

and when ground to pass a 16 mesh screen ($\frac{1}{32}$ inch diameter hole):—

	Per Cent.
From 16 to 100 mesh (granular)...	82½
Through 100 mesh (dust)...	17½

Mr. Krom's opinion confirms Mr. Sahlin's remark that the percentage of dust in the sample of the ore operated on by him was too low, owing to the removal of the dust previously by the exhaust fan at the works.

Crushing the same ore with rolls to pass a 10 mesh screen Mr. Krom found:—

	Per Cent.
From 10 to 100 mesh (granular)	96¼
Through 100 mesh (dust)...	3¾

and when crushed to pass a 16 mesh screen this sample gave:—

	Per Cent.
From 16 to 100 mesh (granular)	93¾
Through 100 mesh...	6¼

These results collectively corroborate the well-known universal experience that grinding dry by attrition produces more dust than crushing with a system of rolls, and that most machines working on the former principle, owing to the high speed necessarily maintained, show greater wear and require more power than rolls do.

The Buchanan granulator is intended to occupy an intermediate place between a large breaker and rolls or other reduction machinery which

completes the crushing of the ore.* The receiving capacity of this granulator is 3½ by 30 inches, or 5 by 30 inches, weighing respectively 10,000 to 12,500 lbs., and have a capacity of 4 to 8 tons per hour reduced to ¼ or ⅜ inch ring-gauge. The machine is intended to reduce the coarse fragments from the jaw-crusher which do not pass the grizzly or screen below the breaker.

The jaws being very wide in proportion to their opening are made double, each being 15 inches in width, and work alternately to reduce the strain on the frame, which is heavy and solidly built. The distance between the jaws which regulates the size of the product can be adjusted by two steel tension-rods which pass through heavy lugs at the top of the frame and through holes cast in a movable tail-block and take the strain. The moving jaw which is pivoted below has an adjustable stroke, as the back toggle bearing can be moved up and down by two hand wheels in a recess cast on the face of the tail-block. The hand wheels pass through yokes or brackets below the tail-block, of which the latter form a part and can be secured by jamb nuts above and below the brackets. If the movable blocks are elevated the angle of the toggles will be increased, but if lowered it will be lessened and the throw of the swing jaw is shortened, preventing packing, whilst making a very fine product. The distance between the jaws and the length of stroke can both be regulated while the machine is in motion. The machine is provided with grease cups fitted with spring covers, a rubber spring and rod to assist the jaw on its back stroke, and the toggle bearings and toggles are chilled at the seats and points to ensure longer wear.

First-motion and Geared Rolls.

With regard to first-motion rolls, Prof. Egleston† says:—"In considering their efficiency and capacity (alluding, of course, to dry-crushing) we must take into consideration that the whole of the surface of the faces of the rolls is fitted to act on the ore, since they are even and parallel and wear so. The ore escapes from the rolls by gravity, just as soon as it is crushed fine enough to fall through the space between them. If the feeding is automatic, or even when it is not, there can be no clogging of them possible."

Theoretically, no doubt, these contentions are all true; but practically the writer does not think the wear can be said to be invariably even and parallel in the two tyres, especially if used for coarse-crushing, and proof

* *Engineering and Mining Journal*, New York, vol. lv., page 323.

† *Metallurgy of Silver, Gold, and Mercury in the United States*, vol. i., page 216.

that they do clog (or what amounts to the same thing) will be given later on. The professor goes on to say:—"A 30 stamp mill, with stamps weighing 850 lbs., dropping 90 times a minute, with shoes and dies 8 inches in diameter, which have in round numbers 50 square inches of surface, will have (50 by 90 by 30 =) 135,000 square inches of surface acting on the ore every minute. It may be considered that two sets of 22 inches rolls will have an average diameter of 21 inches. If these rolls make 80 revolutions per minute and are 14 inches face, they will have a contact surface of 141,120 square inches per minute, or a little more than a 30 stamp mill. If the diameter is increased to 30 inches, taking the average as 28 inches, and the length of face as 16 inches, and the number of revolutions at 80, the surface capacity will be equal to 47 stamps, and at 100 revolutions to 60 stamps. These calculations are made on the supposition that the surfaces acting are of equal efficiency in both stamps and rolls, and for that reason more favourable to the stamps than actual practice shows. With rolls properly constructed the pressure is constant at each instant of time,* while with stamps, on account of the varying height of the ore in the mortar, the cushioning of the stamp against the ore, and the fact that the stamp must not only crush but also force the ore through the screens,† it never can be constant. That the work can be done as well with rolls, which are very compact and take up only a little space, as with stamps, which occupy a much larger one, is shown by the experience of the mills where they are used. That they can crush as fine, is shown by the fact that rolls are now being used to crush after the stamps."‡

The condition of the ore particles leaving the rolls is such, being mostly angular, that to reduce the whole of the stone to the same absolute degree of fineness, it would be crushed with stamps through a finer mesh than the screen of the rolls taken for comparison.

When a movable pillow-block is used to support one of the rolls to prevent any pounding action, and keep them the right distance apart, they should be tied together with tension-rods and spiral-springs, the sliding roll being hinged so that it can swing back freely if anything passes between the tyres and overcomes the strain of the spiral-springs,

* This is only true, the writer thinks, so long as the space between the tyres is absolutely and exactly constant, which is not the case when they are subjected to uneven wear.

† In this connexion the action of the water in washing the ore out should not be overlooked. See also Prof. Munroe's fourth contention, page 344.

‡ It is not less true, however, that stamps are almost invariably used to crush after rolls in Germany, where theory, if anywhere, is carried to its finest point.

but their construction should be solid enough to prevent much vibration. The springs should not bear on the main tension bolts, as in such cases considerable force would be required to adjust the rolls. It is best to enclose the springs, therefore, in a nest between two washers, and to draw these washers together with special bolts, so that the whole will act as a rigid inelastic block until the maximum straining pressure is reached, when, for the first time, full compression will take place and relieve the rolls.

A set of 26 by 15 inches rolls, driven at 100 revolutions per minute, will reduce 20 to 25 tons of magnetic ore from $\frac{5}{8}$ inch to 10 mesh per hour. To hold rolls running at this rate it is absolutely necessary to use springs, as the old-fashioned lever and counterweights would be constantly on the jump. The springs before leaving the shop are compressed with their washers to a strain of 15 tons per square inch. The self-contained box makes it possible for a man with a small wrench to adjust them to any degree of fineness. The tyres should be seated on a conical core, on which they are drawn tight by jamb-bolts. To promote evenness of wear, the narrower the rolls are within reasonable limits the better, and a high-speeded narrow roll of large diameter in this respect is superior to a slow-speeded long roll of small diameter.

The most important part of a set of rolls are their tyres. Many makers use chilled iron or high carbon steel, but the writer's experience is in favour of using rather soft forged or rolled open-hearth steel.

The capacity of rolls is theoretically proportional to the superficial area of crushing surface passing the line of contact in a unit of time, and Mr. Sahlin is of opinion that a peripheral speed of 600 to 700 feet per minute will be found practicable. The writer, however, would suggest that if the diameter of the rolls is not proportioned to the speed the ore is broken partly by concussion, and not carried round and crushed by steady pressure, as it ought to be.

It is important that the axle of movable and rigid rolls should never become oblique under any application of strain, and it is a good arrangement to attach the movable journal boxes to a pair of strong levers firmly keyed into a heavy shaft, so that any excessive strain on one will cause both bearings to yield uniformly. The bearings of rolls should be large, well protected from dust, and perfectly lubricated.

For driving rolls at high speeds belts and pulleys are preferable to the use of geared wheels; the latter wear rapidly, and are rigid when in motion. To avoid slipping, both rolls should be driven by belts, but a greater amount of power may be advantageously applied to the rigid roll.

Adjustable scrapers set on the apron of the feeder should be employed to regulate the feed of the ore at any given point of the face of the rolls, as this is the best means of keeping the tyres true. A shaking trough or overfall flanged pulley are probably the best types of machine to employ as feeders.

Elevators show wear more rapidly than any other part of a crushing plant. Bucket elevators are to be avoided as much as possible; link-belt iron chains and screw conveyors have both proved decided failures for handling large quantities of ore. Leather or rubber belts to which steel or malleable iron buckets are riveted are more durable, but when practicable the best course seems to be to use horizontal or inclined carrying belts, travelling over concave carrying rollers, for elevating and conveying; a system largely in use in the corn warehouses of Liverpool, where the writer has seen it in operation. The greatest inclination at which a belt will effectively elevate is, however, only 27 to 30 degs., and as a consequence great length of belt is requisite to obtain high elevations.

Screening surfaces should err on the side of ample, in fact excessive, capacity. They are also exposed to a considerable amount of wear, and it is therefore necessary to construct them so that the screen plates can be readily changed without removing the frames. In dry crushing a hexagonal prismatic shape is, in the opinion of Mr. Sahlin, the best. The screen shaft should be placed in adjustable bearings, so that the time required to screen a given amount of ore can be regulated to the demands of the rest of the plant. Mr. Sahlin is in favour of using perforated steel plates with oblong diagonal slots fastened by wire-staples to the inside of rectangular hard wood frames, which can be readily clamped on to the cast iron frame. A better plan perhaps is to slide them into grooves in the ends of the arms. The rectangular shape of the frames admits of their being reversed before the nearest end of the screen, which suffers most wear, is cut through, increased service being thus obtained from a set of plates, and the oblong shape of the slots allows foliated crystals of minerals like hornblende and mica, on which the rolls make little impression, to pass the screen without clogging it. Woven wire screens have a tendency to clog, and their durability is less than steel-plate.

The hoppers below the screens should be lined with sheet-iron, or light bars of flat iron set at intervals (the spaces between the bars soon get filled with fine ore, which protects the bars as well as the wood casing).

In guaging the power required to run machinery very variable results may, as before pointed out, be found under different circumstances, and therefore investigations of the sort should extend over considerable periods of time.

With machinery running on a certain class of ore, its efficiency will apparently vary at times as much as 50 per cent., simply through slight changes of velocity, speed of feeding and discharge, or size of the material fed and delivered to the crushers.

Mr. Stetefeldt, in a paper on improved processes for the lixiviation of silver-ores, draws a comparison between Krom rolls and a stamp-battery. He states that the application of rolls is most favourable when the character of the ore permits comparatively coarse-crushing, without interfering with good roasting.

Assuming the crushing capacity of two sets of 26 inches Krom rolls to be equal to a 30 stamp-battery, with stamps of 850 lbs., dropping 7 to 8 inches, 94 times per minute. With two sets of rolls 100 tons of ore can be crushed in 24 hours,* all passed through a 16 inches wire-mesh, consuming not over 4 cords of wood. The fuel required for running 30 stamps is about 6 cords in 24 hours. Assume the following prices:—Freight, 1½d. per lb.; lumber, £10 8s. 4d. per 1,000 feet; wood, £1 5s per cord; carpenters, 18s. 9d. per day; and millwrights, £1 5s. per day. Certain items of construction—conveyors, elevators, screens, and dust-chambers—are about the same as in a dry-crushing mill. The building, however, will be smaller than for a battery, saving not less than £312 10s. in construction. Finally, the rolls requiring less power, a saving of at least £260 8s. 4d. will be made in purchasing and erecting engines and boilers in a mill with rolls.

A 30 stamp-battery, the plant including hardwood screen-frames and guides, wooden pulleys on cam-shafts, Tulloch feeders with iron-hoppers, and all necessary bolts, weighs 90,600 lbs., and costs in Chicago £1,218 15s. According to a statement of Messrs. Fraser and Chalmers, the framing takes about 36,000 feet of lumber, and the expense of setting up is reckoned at £833 6s. 8d. Hence the total cost of erecting a 30 stamp-battery, as specified, is:—

	£	s.	d.
Plant and foundry	1,218	15	0
Freight	566	5	0
Lumber	375	0	0
Erection	833	6	8
	£2,993	6	8
Extra cost of building	312	10	0
Extra cost of engines and boilers	260	8	4
	£3,566	5	0

* Here again a great deal depends on the character of the rock and nature of the ore.

The weight of one set of 26 inches rolls is 12,000 lbs., and costs in New York £468 15s. There is only one self-feeder required, and its weight is estimated at 2,000 lbs., and cost £41 13s. 4d. From these figures we deduce the cost of erecting two sets of 26 inches Krom rolls:—

	£	s.	d.
Plant at foundry	979	3	4
Freight	162	10	0
Cost of erection, including lumber...	145	16	8
	£1,287	10	0

The wear and tear of two sets of tyres, calculated on 20,000 tons of stone, crushed at the Geddes and Bertrand mill, taking the full capacity of the rolls at 100 tons, passed through a No. 16 screen, is estimated as follows:—

	£	s.	d.
Two sets of steel tyres at New York	159	3	4
Freight on 3,264 lbs., at 1½d. ...	20	8	4
	£179	11	8

Wear and tear per day of 24 hours:—

	£	s.	d.
Steel tyres	0	14	4½
Other parts (screens, supplies, and lubricants)	0	7	3½
Wages for repairs	0	5	2½
	£1	6	10½

Mr. Stetefeldt arrives at the following figures of cost for wear and tear of stamps, in a 30 head battery per day of 24 hours:—

	£	s.	d.
In all parts subject to wear and tear, supplies, screens, and lubricants*...	2	7	11
Wages	1	2	11
Total cost of wear and tear of stamps	3	10	10
Wear and tear of rolls...	1	6	10½
Showing a difference in favour of rolls of	£2	3	11½

In comparing expenses of running rolls and stamps, the interest and redemption of the excess capital, required in the construction of the original plant for stamps, cannot be neglected. Considering the short life of most silver-mines, this item should not be taken at a lower rate than 15 per cent. per annum. If we take the running time of the mill at 350 days in a year,† and consider a mill with stamps will cost £2,278 15s. more than one with rolls, the interest and redemption amount to 19s. 6d. per diem.

* Subdivided as follows:—Shoes and dies, 40 per cent.; tappets, boxes, cams, stems, cam-shafts, fingers, and boxes, 38 per cent.; screens, lubricants, screen-frames, guides, and carpenters' and machinists' supplies, 22 per cent.

† In Nevada they work on Sundays as well as on weekdays, or used to do so, though the practice is certainly inexpedient and undesirable.

Summarizing the above, we find the following daily saving by using 26 inches rolls, as compared with stamps :—

	£	s.	d.
Wear and tear and repairs	2	3	11½
Interest and redemption of capital ...	0	19	6
Fuel, 2 cords at £1 5s.	2	10	0
Total	£5	13	5½

Admitting the correctness of some of these figures, when applied to different cases, for example, supposing a stamp-mill to be run by water-power (as is often the case), the last item would obviously have to be excluded, and thus the advantage to the credit of rolls would at once be reduced to £3 3s. 5½d.

Then, again, as regards the extra cost of the buildings, the writer scarcely thinks the comparison altogether fair towards stamps. If you take merely the stamps and rolls, without the pans, settlers, furnaces, leaching-vats, and other apparatus necessary for saving the silver, it may hold good, but his own opinion is (till the contrary has been proved by actual figures), that if these are included, the tendency towards cost of building is precisely the other way.

The supposed saving can only be in width of building, but if it is a dry-silver amalgamating-plant, the room occupied by the pans will involve the same space cross-ways in the pan-room, whether rolls or stamps are used, and if, as Prof. Egleston remarks,* "the rolls are usually arranged in sets of two [as they ought to be and frequently are]. One of these receives the coarse ore from the crusher, which, after passing them, falls on a screen of determined size. What does not pass goes to a second pair;" it will add very considerably to the height of the mill, an addition which can be proved to be proportionately far more costly, than a saving in width.

Comparing the plans of some lixiviation-mills with those of dry-crushing pan-mills, it almost appears as if the latter class of building, as regards framing, was the more elaborate of the two. Take for examples the arrangement of a lixiviation-plant, illustrated in Mr. Stetefeldt's work on lixiviation, and a dry-concentrating roller-mill, in Galena Cañon, Nevada. This latter (speaking from personal acquaintance with it), the author ventures to say, is an extremely elaborate structure indeed, far more so (owing to its great height) than any pan-mill he has ever seen. A section of this dry-crushing plant is given by Dr. Raymond in one of the United States Government Reports.

* *Metallurgy of Silver, Gold, and Mercury in the United States*, vol. i., page 215.

Prof. Egleston adds to his previous statement quoted above this noteworthy remark: "When very fine crushing is required, three pairs of rolls can be used; but generally two sets are sufficient." This means that, in some cases, you can double, in others almost treble, the cost of plant at the foundry, and count on freight for four to six sets of rolls instead of two, whilst making a considerable addition as well to Mr. Stetefeldt's estimate for the wear and tear on the two or four additional sets of tyres, and check-plates, as the case may be. Under such circumstances, the saving of £2 3s. 11½d. and 19s. 6d. for wear and tear, and interest and redemption of capital, may be found to be really negative quantities.

Mr. Munroe gives the wear and tear of eight pairs of Cornish rolls, 14 by 30 inches, at Bonne Terre, after crushing 224,303 tons of magnesian limestone lead-ore in 1886-87 as under:—

	£	s.	d.
Seven pairs of chilled tyres, at £25	175	0	0
Gear-wheels and pinions	69	15	10
Total	£244	15	10

About £30 12s. 0d. per pair per annum. The tyres of the rolls, he remarks, used for coarse-crushing, are not turned down when worn, but are replaced by new ones, a statement which would indicate that there is more waste of metal in the use of rolls than stamps, and the author has frequently noticed the worn-out shells of rolls lying about the yards of works. It is owing, he presumes, to their considerable bulk and weight, that it does not pay (even if of iron) to break them up or dispose of them as they stand. The tyres of the fine-rolls at the Bonne Terre works are kept in good condition. Dr. Peters remarks,* "That the hardest chilled iron may be turned with an ordinary tool without difficulty, if a sufficiently slow motion is made use of in the process." In this way a set of chilled iron shells may be preserved in condition for fine crushing for a much longer period than usual.

For the screens connected with the rolls at Bonne Terre 21 sets of perforated plates, at £12 13s. 1½d. = £265 15s. 7½d., were required, or an average of 2·6 sets per year for each pair of rolls.

Each set of steel tyres on the rolls, therefore, lasts about a year, and a set of screen-plates about four months.

Prof. Egleston, again speaking of the experience at the Geddes and Bertrand mill,† states that the ore was crushed so as to pass a 15 to 20

* *Modern American Methods of Copper Smelting.*

† *The Metallurgy of Silver, Gold, and Mercury in the United States*, vol. i., page 487.

Sumr
26 inch

372

mesh screen; 80 mesh screens were first used, but it was found that the material did not discharge from these as well as from a coarser mesh, and that there was no necessity for treating the ore finer, as the roasting and leaching were better done on the coarser ore. Taken in connexion with what has been said before, the writer thinks this proves one of the points that he has emphasised—that rolls are not adapted to reduce ore in one operation to fine size. And this explains why they are not suitable for dry-pan amalgamation, because it means that either two (if not three) pairs of rolls must be used, one below the other, or extra work in the shape of grinding will be thrown on the pans. Added to this, there is the danger of a complete stoppage of a whole section of the works for a time, should a serious breakdown occur, and the time lost in changing tyres is not so very much less in the aggregate than changing stamp-shoes, as the latter is generally done when cleaning-up.

Messrs. McDermott and Duffield remark*: "Of late years rolls have been advocated for fine-crushing, and where the operation has to be performed dry, as a preparation for roasting with leaching, or chlorination, there is no question of the advantages to be gained.† For ores requiring wet treatment, however, free gold and silver and concentrating-ores, there is nothing but disadvantages connected with the use of rolls. Although there are various devices for concentrating and amalgamating dry-pulp, it is quite certain, from the failure of these processes in practice, though known for many years, that, where water is obtainable, dry treatment is not worth considering. The mere item of drying the ore, and the disadvantage of dust in dry-crushing—even if the process be granted the full efficiency of equivalent wet methods—are sufficient to prevent any serious consideration of dry-treatment, except as a matter of necessity from the absence of water. No one desires to dry an ore, crush dry, and then wet it again. For dry-crushing, stamps are not nearly so effective as for wet work, and, therefore, what would be a favourable comparison for rolls in one case becomes very different in the other. Rolls require screening and re-screening, with steady return of uncrushed particles, and all this means excess of water introduced, and the running of screens and elevators in addition to the crushing machinery. Again, the surface of roller-shells must be kept turned up true to do effective work in fine-crushing, or their capacity drops directly, while the capacity of stamps varies very little with wear of castings, as increased drop tends to neutralize

* *Gold Amalgamation and Concentration*, page 78.

† There is an exception to this mentioned by the writer, *Trans. Fed. Inst.*, vol. v., page 135.

decrease in weight. Some ores containing clay will press into thin cakes by the rolls, and become an increasing factor in the return from the screens."

The writer is not going however to deny, that under certain conditions, rolls are more suitable to employ than stamps, quite the reverse, as he has previously pointed out that rolls have most marked advantages over stamps, for certain special purposes. Nothing, in fact, too strong can be said in their favour, if it be intended to reduce an ore gradually in size, to an even grade, of coarse or fine sharp sand (1 millimetre or 0·039 inch and over), with a minimum production of slimes; and the whole method of Continental dressing, rests on this principle.

To obtain the mineral, in fact, with a minimum loss from comminution, the ore must be broken up, only just enough to unlock all the mineral it may contain, down to a certain size, and the mineral thus freed, must be separated from the mass at once; now whilst rolls naturally lend themselves to this class of treatment, stamps do exactly the reverse. On this account and others that have been pointed out, rolls find a wide application, in coarse-concentration, and lixiviation, in both of which evenness of product is essential for good work, and in sampling-works, attached to smelters. In fact, in most instances where one has to deal with the reduction of heavy pyritic ores, they justly hold the field, and in exceptional cases can be used in chlorination-works, though it is the more usual practice to treat free-gold ores in a stamp-battery.

Passing over the unusual case of the native copper amygdaloid, of Lake Superior, for which steam stamps are found best adapted, the author ventures to lay down four cases for which rolls are generally unsuited :—

1. Where you have to deal with a concentrating-ore like some grades of German lead-ore or Cornish capel, in which the mineral is finely disseminated (German, *eingesprengt*) or sprinkled through the gangue, it can be better liberated, either by stamps or roller-mills.
2. A clayey ore.
3. Any ore which requires, for reasons other than the one first stated, to be reduced in one operation, from a comparatively large lump-size, to fine powder.
4. Where amalgamation of any kind is to follow directly after comminution, especially when large quantities of ore carrying finely disseminated free-gold, have to be manipulated wet.

The writer will not comment on the first and third cases, as he has already referred to them. With regard to a clayey ore he alludes, of

course, to an ore containing a considerable proportion of clay, like some gold, zinc, and iron-ores, in which case the difficulty can only be overcome by passing the stone through a special cleansing-apparatus, generally a conical revolving-drum, 15 to 18 feet long, and 6 to 12 feet in diameter, fitted sometimes with inside blades, which break up the clay, allowing it to be washed off by a strong stream of water playing on the inside of the machine.

If the ore carries only a little clay, and the stone comes to the mill dry, it does not matter so much; if damp, it is advisable to run the rolls and screens wet, using a water-spray as they do at Clausthal. Whilst an excess of water does no harm but rather good, a little moisture in the stone itself, say 3 to 5 per cent., produces the effect of making it sticky, and is most pernicious.

In regard to the fourth case, the writer has already endeavoured to show the unsuitability of rolls for dry silver-amalgamation, and if this be granted, they are doubly so in most instances, for a wet free-gold process,* as it means the addition to the plant not only of elevators, screens, etc., but mechanical agitating and amalgamating-machinery as well, which, when the practical question of ultimate cost comes to be considered, reverses every other advantage to be gained. Added to this, no mechanical agitator, the writer believes, has yet been devised, which will give the natural splash of the stamps, which is as essential to plate-amalgamation as the turn of the wrist is to panning a gold prospect properly.

The writer may instance in proof of his contention what happened at the Haile mine, South Carolina, citing as his authority a paper by Mr. Theo. Blake.† The ore is a tough quartzite, carrying a large percentage of sulphides, which, according to Messrs. A. Thies and Wm. B. Phillips,‡ is associated with talcose slates. Whatever free-gold there is, is extremely fine, and is got out now by stamping in a battery, provided with inside and outside copper-plates. The pulp, which contains the balance of the precious metals present, is thence led on to Embrey end-shaking tables, and the concentrates thus obtained (representing about 9 per cent. of the ore) are worked, as previously described, by the Thies barrel-chlorination

* The question may be asked how this statement is reconciled with the Pestarena practice. The author may point out that the conditions of that particular case, like those of Mount Morgan, Queensland, are unusual, since the gold in the pyrites appears to be unusually free, the stone is heavily mineralized, and is subjected to a certain amount of concentration previous to crushing, a most unusual procedure, which is only practicable, owing to the low price of labour in Italy.

† *Trans. Am. Inst. Min. Eng.*, vol. xvi., page 753.

‡ *Ibid.*, vol. xix., page 601.

process, which appears to have at length solved this much-vexed question. The history of the undertaking is a peculiarly instructive one. Rock-breakers, rolls, and amalgamators, coarse and fine rock-breakers, and Chilian mills, the Disignole process, and dry-crushing, having each, the writer believes, had their turn, and been proved previously unprofitable.

Prior to 1888, it appears that the plan adopted was to pass the ore through two Blake Challenge stone-breakers (20 inches by 10 inches and 30 inches by 5 inches) and two multiple-jaw crushers, and thence through two sets of 30 inches by 16 inches Krom swinging-block rolls, from which it passed over Atwood amalgamators, and finally the pyrites was concentrated on Embrey tables.

Now, according to Mr. Blake,* who appears to have taken the matter up about this time, large as the capacity of the Krom rolls was, they could not pass the material they were designed to handle dry (from ½ inch to 40 mesh) through quick enough, which would appear to be a mechanical indictment against the rolls: that it is not so, is due to the point already noticed, that two sets of rolls, one coarse and one fine, ought to have been used to give them quite fairplay. In fact if rolls require constant attention, and the ore does not pass through them properly crushed, you might for all practical purposes it seems, just as economically employ two sets at once, as incur the extra expense of wear and tear, coupled with the necessity of screening, re-screening, and elevating, the partially crushed products. If we follow Mr. Blake a step farther, he tells us, moreover, that they were running these rolls at 100 revolutions per minute, and this in itself was sufficient, the writer thinks, to account for their failure, to meet the call made upon them.

With coarse-crushing rolls general experience goes to prove that, in order to secure high tangential speed with a view to obtain large capacity, the diameter of the rolls must be duly proportioned to their rotatory velocity. This would be accentuated the greater the relative difference in size between the uncrushed and finished products, which in this case was extremely large. If speeded unduly high, the rolls will in fact refuse to bite ore-particles above a certain size, and you have as a result, a layer of ore between the rolls, dancing about like peas on a drum; in which case speed will produce just the opposite effect to that intended, and tend to choke them. The same also is true of a clayey ore, as a great deal of it will, in that case, be drawn through them in clots, causing extreme and uneven wear, and an enormous increase in returns of only partially crushed ore, so that an undue percentage has to pass through them again.

* "The Blake System of Fine Crushing and its Economic Results." *Trans. Am. Inst. Min. Eng.*, vol. xvi., page 753.

Mr. Kunhardt cites a case,* which actually shows that rotatory speed increases, it is true, the capacity of the rolls up to a certain point, but cannot be carried beyond given limits.

At one mill, where careful trials were made on the Continent, a pair of rolls 25½ inches in diameter and 11 inches face, required to run at 26 revolutions per minute in order to crush 15 tons of hard quartzose ore per hour, from 2 inches size to ⅝ inch. Reducing their speed diminished their output, but it was found, that without constant attendance, to shove the ore down between them (a remedy worse than the disease), the maximum speed they could economically be run at, was 16 revolutions per minute. The same difficulty, though to a less extent, may happen in fine crushing, and will be largely influenced by the difference in size between the ore fed to the rolls, and the size of the product they are required to deliver. The effect at Haile of running the rolls in the manner described was in fact precisely what one would expect. The belts constantly slipped, or were thrown, bringing the rolls suddenly to a full stop, and the surface of the shells or tyres had, in crushing 500 tons each, worn down ¼ inch, and become pitted and corrugated, while the wear on the screen-cloths actually amounted to 4s. 2d. per ton.

Mechanically, then, it was not surprising that the rolls jibbed, but that in no way alters the writer's proposition, viz.: that rolls are unsuited to wet gold-extraction, as a general rule; in fact, subsequent experience at Haile strengthens it, since they were superseded by a couple of Chilian mills, which Mr. Blake claims afforded a full solution of this difficulty, giving a capacity of 3¾ tons† per hour each (crushing the toughest quartzite of the Blauvelt mine), and as much as 10 tons of the softer decomposed ores, with a consumption of about 4 cords of wood per diem. The ore was crushed wet in this way. The two wheels of the mills each weighed about 1 ton, and were 4 feet in diameter, with 8 inches face, making 40 revolutions per minute. The wear of the tyres and segmental dies, estimated on 3,000 tons of ore crushed, was 0·12 of a pound per ton, and the cost of screens (screening the ore wet in a sort of raff-wheel) was reduced to (10 cents.) 5d. per ton. Yet, notwithstanding the question of capacity being solved, even this arrangement, was (it would appear from its abandonment) found too expensive as compared with stamps, for

* *School of Mines Quarterly*, series 2.

† At Pestarena, in Italy, Messrs. J. Taylor & Sons state that a Marsden 14 inches stone-breaker and Cornish rolls, 20 by 13 inches, are capable of passing through a sieve of 9 holes to the square inch, 1,000 tons of ore per month, working 10 hours per day, and 6 days per week; representing about this same quantity, 3¼ tons per hour, but crushing it much coarser.

direct amalgamation of the ore in bulk; and as a result, the final change back to stamps had to be accepted as a commercial necessity.

In many low grade mines an increase in the tonnage crushed with an existing plant means not only a lower cost per ton of ore, but an increased revenue, which in some cases may make a profitless mine a success, and in certain cases, therefore, rolls may be profitably used for coarse-crushing before stamping it. The only mill constructed on this principle is the Huanchaca mill, at Antofagasta, Chili. By this means an increase of at least 20 per cent. in the capacity should be secured. It would not be out of the way either to have fine screens below the coarse trommels, so that the finer portion (no inconsiderable amount with many ores) might be sent direct to the pans or furnaces without passing through the battery, when the arrangement of the mill would admit of this, as was done with marked advantage at the Hermosa mill, Harshaw, Arizona.*

The ore should not, however, in such an arrangement be crushed smaller than $\frac{1}{4}$ inch by the rolls, and $\frac{3}{8}$ inch would be better, to avoid danger of choking the battery. Such an application of rolls might be utilized advantageously. Then again, dry-crushing by this means will always hold its own in lixiviation and some cyanide plants, because when the ore is crushed wet it must either be run into settling-tubs or direct to the leaching vats. In the former case extra labour is required, whilst in the latter the ore is deposited in layers according to specific gravity. The heavier and more rebellious minerals fall to the bottom, and the lighter particles forms a coating on the top which is almost impervious to the solution, and is consequently harmful. Finally, in connexion with dressing-machinery, rolls will never be superceded, but when the former is of a certain fixed capacity, and in direct connexion with them, it is a mistake for the crushing capacity of the rolls to be in excess. Should it be so, instead of gaining time and saving money by putting the ore through quickly, heavy loss is sustained by producing poor quality concentrates, or mineral is lost in the tailings by unduly forcing the capacity of the rest of the plant. A difficulty only to be got over by having storage bins, which in many instances could not be introduced without incurring unnecessary expense for re-handling or destroying the automatic sequence of operations which ought to obtain in all well-designed plants.

Bullion Refining, and Sampling.

When gold, containing iron (such as the crude bullion from mill-retorts), is being refined at the United States mints, Mr. R. P. Andrews†

* *Trans. Fed. Inst.*, vol. v., page 288.

† *Engineering and Mining Journal* (New York), vol. lv., page 54.

states that it is the practice to melt it in fire-clay crucibles placed in plumbago pots, with sulphur and carbonates of soda and potash, refining it afterwards with nitre. The surface of the melt is covered with a layer of iron sulphide slag, which may be removed by nitre, but much time will be gained if the melt is poured off into a mould, allowed to cool, and the slag broken off. This slag should be remelted with iron scrap for the recovery of any contained gold; the bar gold is remelted and any gold obtained from the slag is added. The first refining is done by the addition of nitre, with occasional additions of borax-glass to thicken the flux. The flux is removed by means of a plumbago stirrer, which must be kept at the same temperature as the furnace. After the iron has been removed the lead and antimony are next got out by the oxidizing action of nitre. If these metals are in excess the successive additions of nitre will have little effect on them for some time. When the action has commenced its progress can be judged by the colours of the flux skimmed off, commencing with whitish yellow it passes to light green and then to dark green. Before the final removal of the lead rainbow colours pass rapidly over the metal as in cupellation. Towards the end of the refining the gold bubbles slightly, but as the last trace of lead disappears the surface becomes quiet and smooth as plate-glass. If the lead and antimony stubbornly resist the action of the nitre a little bichloride of mercury or sal ammoniac may be added. The chloride penetrates the whole body of the metal and forms volatile chlorides which rapidly rise and pass away. This action is different from that of nitre, as the latter remains on top of the metal and oxidizes the lead and antimony as they rise. Bichloride of mercury is more certain in its action than sal ammoniac, but it is more dangerous and therefore hardly to be recommended. Some refiners add oxide of copper to aid the extraction of lead, arsenic is removed by the same means as lead and antimony, but more time and close examination are necessary to get rid of it.

In the case of very base bullion, cupellation is almost a necessity. It can be effected in a wind-furnace connected by a wide throat to the flue of the other furnaces. The bottom of a plumbago melting-pot, well tamped with bone-ash, and hollowed out to receive the melt, will form a first-rate cupel. The cupellation requires no other attention than to keep the melt from freezing, and the process can proceed while other work is going on. If the metal should solidify before the impurities are all removed, their final removal can be effected by crucible refining.

Nitrate of potash is usually used in refining, though it contains 10 per cent. less nitric acid than nitrate of soda, the latter, however, is generally

impure and contains sodium chloride which, in the refining process, causes spurting and consequent loss of gold.

If the operator employs a plumbago pot for refining, the corrosive effects of nitre on the plumbago can be prevented by lining it with a bone-ash ring, which will gradually absorb caustic potash and become well fixed to the inside of the pot. This lining will protect the crucible and keep the nitre in the central part of the surface of the melt.

Silver is refined in the same way as gold, but extreme care must be taken to keep the temperature of the furnace low or silver will be volatilized. It is not advisable to use either bichloride of mercury or sal ammoniac in refining this metal.

The writer has endeavoured in this paper to steer clear of prejudice for or against any special make of machine or process, and wherever possible to express himself in general terms.

An engineer has obviously no more right to experiment in new machinery and methods of treatment on his own account with his employer's capital, without distinct permission to do so, than he has to accept a commission on the purchase of a machine as an inducement to recommend it. At the same time, be it said, the mine-owner not infrequently pursues the short-sighted policy of restricting the engineer in the matter of independent investigation, regardless of the loss to himself, and even goes so far as to offer direct inducement to the maker to produce inferior cheap work, by showing a total disregard of its quality, selecting plant sometimes on his own initiative, simply because it happens to be cheap. The author believes this interference is a most frequent cause of the failure of many mining undertakings, which is not to be wondered at, where the selection of plant and process is perhaps left to a person who has never seen a mine in his life, or at any rate knows nothing of the practical requirements of a business, so comprehensive in its nature, that a life devoted to it, and nothing else, is scarcely long enough to master more than its rudiments. It is undoubtedly to the interest of the mine-owner and maker alike, that the choice of mining-plant should be left to the engineer, who is responsible for erecting and running it; at any rate, the person who assumes such a responsibility, should be personally acquainted with the mine and its local surroundings, in order to form a judgment of any value, upon such a matter.

It is the more important, because before you can alter a process that has once been established, however defective it may be, you must see your way to effect a saving which will compensate for the loss of capital sunk

in a first venture; in fact, the sum of the annual dividends everybody expects, and ought to get, when they put their money into a mine, largely depends on making a correct first choice, in selecting the means of treating the ore.

A battery improperly built, from a desire to do the work cheaply or carelessly, cannot be run at a high speed, but when well put together it may be run for a long time without damaging a bolt. The extra prime cost of a mill so constructed is a slight consideration if, as frequently happens, it enables a low grade ore to be treated at a profit which would otherwise be worked at a loss, as in the latter case a large capital sum is thrown away which could have been profitably employed by but a slight additional expenditure.

An engineer has to look at the business facts as he finds them; he is liable to make mistakes, probably, like other people, in doubtful cases, but with ordinary experience, judgment, and business prudence, he should be able to select the proper machinery for any particular purpose, such as will best forward the interests of those he serves—a fact mining companies might do well to consider more than they do.

Miners have a great deal to learn regarding the comparative merits of different structural materials used in machinery, applied to different purposes, the behaviour and comparative value, for instance, of mottled-iron (a mixture of 85 parts of hardest white-iron and 15 parts of tough grey-iron), ordinary chilled white-iron,* and ferro-alumina (a highly crystalline hard white-iron) for stamp-shoes and dies, as compared with ordinary cast-steel, chrome-steel, nickel-steel, aluminium-steel, and manganese-steel, or Hadfield steel, and for such a purpose evenness of wear, which involves uniformity of casting, is, the author thinks, rather to be aimed at than extreme hardness; or, to take another case, the comparative merits of open-hearth steel, which, though it costs about 5 per cent. more, is said to be a better material for roll-tyres than Bessemer-metal. Ordinary cast-steel for stamp-shoes lasts longer than iron, but costs proportionately more; on the other hand, a less frequent adjustment of drop is required with the more slowly-wearing material. In other cases it appears to be advantageous to use steel shoes and iron dies, as is done in some of the Californian mills, whilst in Australia excellent results are obtained with cast-iron shoes and wrought-iron dies. In districts remote from railways, and where freight charges are high, the first cost of material becomes, too, a matter of less moment than its durability.

* White-iron heads with a malleable-iron shank are much used. In dry-crushing wood wedges to fix the heads on with are apt to get loose, and the shoe drops off, they are therefore replaced by an iron strap ¼ inch thick.

If, for instance, it requires two sets of iron shoes and dies to do the work of one set of steel ones, it will evidently cost double as much in freight to keep the mill supplied with the former, and what would be the cheaper material at the foundry becomes the most expensive perhaps to maintain in use at the place where it is employed. This is the reason why chrome-steel is preferred at mines like the Mammoth in Pinal county, Arizona.

In many districts local foundries will recast worn-out iron shoes and dies, but are unable to produce steel-castings, and there is no market for old steel, while scrap-iron can generally be readily disposed of.

Speaking generally, it may be said that the wear of shoes and dies in a battery is affected considerably by the depth of discharge; and the use of rock-breakers, grizzlies, and automatic feeders, by promoting even feeding, tend to produce evenness of wear. The best results are to be got not so much from the hardest material as by having the die made of metal more tough and less brittle than the die, examples of which have been given, or, to take a still simpler case, chilled and unchilled iron are also sometimes used. The wear of shoes, made of various materials, varies from 3·6 to 21·3 ounces of metal per ton of ore crushed, while that of the die has a minimum of 3 ounces and a maximum of 7·9 ounces. In the matter of cost the lowest is at the rate of 1d. and the greatest 3·8d. per ton of ore for the shoe, whilst for the die it runs from 2·8d. to 0·3d. The combined cost varies from 2·0d. to 6·5d. See Appendix A.

Again, the engineer requires to ascertain the relative quality of various kinds of timber, lubricants, etc., and in fact of supplies of all kinds, to compare their serviceability for different purposes, with their cost, information which is well worth collecting. Neither must he neglect improvements in tools, and small labour-saving appliances, which are constantly being introduced, tending to lighten the mechanical labour of the staff of the mine and works, and conduce to better or more expeditious, and consequently cheaper work.

It is a knowledge of the little wrinkles which save a few seconds time, not less than those which will save a penny on a purchase, or an extra fraction of a grain of the precious metals, which determine the economy or otherwise of management. New inventions and discoveries will no doubt be made in mechanics and metallurgy, and the dynamical applications of electricity will, at no distant date, have a most important bearing upon cheapening the cost of mining in districts where water-power is available as a prime motor, especially so in mountainous districts (like the Pyrenees for example) where local transport is of course difficult.

The author thinks, however, we must look for advancement in the future rather towards perfecting the details of the machinery and processes we at present possess, than to the discovery of new principles. We might substitute, for example, a small motor of the Diehl type for the fly-wheel belt and pulley of a rock-breaker or other machine, and drive it direct in this way perhaps with advantage in some situations.

Perfection is as unattainable in milling as in any other branch of industrial art, otherwise progress would cease. We can compare old methods with new not only with complete satisfaction at the advancement made, but with the consciousness that where so much improvement was possible, much room for improvement may remain.

The author regrets that he has recorded little that is new; he has drawn on scattered sources of information, and on the knowledge of others, viewed from the standpoint of personal experience, which is the only way he thinks to press forward towards a knowledge of the science constituting the important calling of the mining engineer, which, for diversity of interest, breadth of scope, and general utility (common features of all engineering), is surpassed by few, if any, of the paths in life that can be followed in this progressive age.

In conclusion, the writer wishes to acknowledge his indebtedness in particular to the *Engineering and Mining Journal* of New York, the writings of Mr. T. A. Rickard, and others for many valuable suggestions, which it is impossible in every case to individually acknowledge, though he has endeavoured to do so as far as lies in his power.

APPENDIX A.—Wear of Shoes and Dies in Stamp Mills.

Name of District.		Metal.	Weight. New.	Weight. Worn out.	Ore Crushed during Time of Service.	Metal Worn per Ton of Ore Crushed.	Cost of the Metal per Lb.	Value of the Scrap.	Cost per Ton of Ore Crushed.	Total Cost per Ton of Ore Crushed.
			Lbs.	Lbs.	Tons.	Ozs.	d.	d.	d.	d.
Gilpin Co., Colorado ..	Shoes	Cast iron ..	83	27	80	11·2	2	½	1·91	2·97*a*
	Dies	Do.	48	26	78	4·5	2	½	1·06	
	Shoes	Chrome steel	111	31	202	6·3	4	..	2·20	3·58
	Dies	Do.	55	25	159	3·0	4	..	1·38	
Grass Valley, California	Shoes	Do.	152	48	251	6·6	4½	½	2·58	4·71*b*
	Dies	Cast iron ..	93	45	96½	7·9	2½	½	2·13	
Angel's Camp, California	Shoes	Chrome steel	175	40	585	3·6	4½	..	1·35	2·03*c*
	Dies	Cast iron ..	95	35	275	3·5	2½	½	0·68	
Mammoth, Arizona .	Shoes	Chrome steel	132	40	190	7·7	5½	..	3·82	6·87*d*
	Dies	Do.	120	37	240	5·6	5½	..	2·75	
Bendigo, Victoria ..	Shoes	Cast iron ..	180	38	115	19·7	1½	½	1·83	2·18*e*
	Dies	Wrought iron	98	26	335	3·4	1½	½	0·35	
Clunes, Do. ..	Shoes	Cast iron ..	196	56	105	21·3	1½	½	2·33	2·77*f*
	Dies	Wrought iron	138	30	420	4·1	1½	½	0·44	
Harrietville, Do. ..	Shoes	Fagot iron ..	172	38	185	11·5	1½	..	1·63	2·36*g*
	Dies	Do. ..	84	37	200	3·7	1½	..	0·73	
The Thames, New Zealand	Shoes	Cast iron ..	170	51	135½	14·1	1½	½	1·70	2·82*h*
	Dies	Do. ..	108	42	141	7·5	1½	½	1·12	

a No rock breakers or automatic feeders, ore moderately soft, lor *g* drop, wear of dies very variable.
b Rock breakers and feeders, ore very hard, dies contain ¼ steel scrap.
c Rock breakers and feeders, ore soft, short drop.
d Ore medium, rock breakers and feeders, no grizzlies.
e No rock breakers or feeders, ore variable, but medium hard.
f No rock breakers, feeders used, ore almost entirely quartzose.
g No rock breaker, feeders used, dies wear very irregularly.
h No rock breakers or feeders, ore of very variable hardness.

APPENDIX B.—Costs of Mills.

Black Hills.

	£	s.	d.
60 stamp gold-mill, containing 3 (15 by 9 inches) Blake crushers, 6 grizzlies 5 by 12 inches, 12 Tulloch feeders, 60 stamps of 850 lbs. each, 12 mortars, 12 electro-silver-plated copper-plates, 2 (30 inches) clean-up pans, 2 (10 inches) retorts, 2 amalgam safes, overhead crane, track, etc. All necessary shafting, pulleys, and belting, 18 by 42 inches Corliss engine, 2 (54 by 16 inches) tubular boilers, 30 inches tubular heater, No. 5 Knowles pump, etc. Approximate cost, erected	15,416	13	4
120 stamp gold-mill, containing 6 crushers, 24 feeders, all copper-plates, 2 (48 inches) clean-up pans, 1 (24 inches) retort, 120 stamps (900 lbs. each), 24 mortars, all shaftings, pulleys, and belting, 6 amalgam safes, 24 by 48 inches Corliss engine, and 4 (54 by 16 inches) boilers. Approximate cost, erected ...	51,041	13	4

Mexico.

40 stamp silver-mill, containing 2 (10 by 7 inches) crushers, 4 grizzlies, 40 stamps (750 lbs.), 8 feeders, 16 (5 feet) pans, 8 (8 feet) settlers, 2 (10 inches) retorts, 8 amalgam safes, 2 clean-up pans, 27 tighteners, all shafting, pulleys, belts, bolts, etc., and engines and boilers, etc. Approximate cost, erected	29,166	13	4
100 stamp gold-mill, containing 4 (15 by 9 inches) Blake crushers, 8 grizzlies, 20 feeders, all copper-plates, 2 (30 inches) clean-up pans, 20 mortars, 100 stamps (850 lbs. each), 22 by 48 inches Corliss engine, 4 (50 by 16 inches) tubular boilers, 1 (30 inches) heater, 1 (3½ by 16 inches) pump, 2 cranes, all shaftings, pulleys, belting, etc. Approximate cost, erected	41,666	13	4

APPENDIX C.—Comparative Table of Stamp Mills.

The following table shows at a glance the general characteristics of six of the principal gold-mining centres described by Mr. T. A. Rickard :—**

District.	Average Contents of the Ore.	Average Milling Costs.	Average Number of Stamps.	Weight of Stamps.	No. of Drops per Minute.	Height of Drop.	Depth of Discharge.	Crushing Capacity per Stamp.	Capacity of Mill.
	Dwts.	s. d.		Lbs.		Inches.	Inches.	Tons.	Tons.
Clunes, Victoria	9	2 5½	60	950	80	8	4½	2½	150
Ballarat	8½	2 4	40	950	70	8	3	2	80
Bendigo	9½	2 5	40	900	72	9	3½	2½	90
Otago, New Zealand	10	2 11	28	800	77	7½	3½	1½	30
The Thames, New Zealand ..	9½	3 11½	30	700	70	8½	2½	1¾	52
Charters Towers, Queensland ..	25	12 6	30	950	74	8	2	2½	75

District.	Variety of Screen.	No. of Holes per Square Inch.	Percentage of Concentrates.	Average Gold Contents.	Retort Percentage.	Fineness of Bullion.	Life of Screens.	Loss of Mercury per Ton of Ore.	Consumption of Water per Stamp per Minute.
			Per Cent.	Ozs.	Per Cent.	Per 1,000.	Days.	Dwts.	Gals.
Clunes, Victoria	Perforated copper-plate ..	100	¼	3	40	970	25	¼	8
Ballarat ..	Punched Russian iron ..	160	2	2	48	963	12	5	5½
Bendigo ..	Punched Russian iron ..	143	1	2½	50	955	11	7½	6½
Otago, New Zealand	Punched Russian iron and steel cloth ..	140	*	*	41	930	7	7	4
The Thames, New Zealand	Punched Russian iron ..	160	†	†	42	665	6	‡	§
Charters Towers, Queensland ..	Perforated charcoal iron ..	225	‖	‖	25	790	5	¶	¶

* Pyrites not saved as a rule. † Very variable; saved by blankets and treated in pans. ‡ Mercury usually attained a loss of 12 to 20 dwts. § Very variable. ‖ Very variable; from 1 to 7 per cent. and 2 to 20 ozs. value. ¶ Differ widely, according to number of pans and concentrators.

APPENDIX D.—Jaw-crushers.

Besides the crushers described in Part I. of this paper, some engineers favour a parallel motion by means of links, but as it is necessary in such cases to greatly increase the weight of the moving parts the vibrations will either become excessive or the speed must be reduced with consequent loss of output. It has been customary in the larger crushers to make the jaws in sections moving alternately. This reduces the vibrations but complicates the construction and increases the wearing parts, both vital defects with machines of the reciprocating jaw type. The writer favours the plan of pivoting the oscillating jaw, below the face plates, on the Dodge and Krom principle. By this arrangement the product is finished very evenly, the travel of the jaws is shortest, and the leverage greatest at the point where most work requires to be done. The working speed of a 10 by 20 inches Krom crusher is 400 revolutions per minute. The swinging jaw is made hollow to decrease its weight, and the face plates are made of rolled steel bars placed edgeways.

** "Gold-milling in Australasia."—*Engineering and Mining Journal* [New York], vol. lvii, page 101.

APPENDIX E.—ERRATA.

On page 16, line 9, *for* " (carrying less than 6 ounces," *read* " (carrying not less than 6 ounces."

On page 20, footnote, *for* " The idea Mr. Pearce intends to convey," *read* " The idea, it appears, Mr. Pearce intends to convey."

On page 25, line 2, *for* " ratio to one another," *read* " ratio to the item for labour."

On page 36, footnote, *for* " with 6 to 8 per cent. solution," *read* " with 0·6 to 0·8 per cent. solution;" and *for* " containing 2 to 4 per cent. of cyanide," *read* " containing 0·2 to 0·4 per cent of cyanide."

On page 40, line 32, *for* " more particularly as regards cost, which precludes," *read* " (more particularly as regards cost) in situations which preclude."

With reference to remarks made on page 50, Mr. Hamilton Smith wrote the author on September 25th, 1893, as follows:—" Brückner's furnaces were first used at the Alaska Treadwell mine, and changed to Spence furnaces, but with results not quite so good, costs being larger with no better extraction. In 1890 the Spence furnaces were abandoned, and long, single hearth, reverberatory furnaces were built, which have since been in use. Their capacity is 4 tons per diem each, allowing for repairs. With the Spence furnace, costs per ton were fully $18, and the gold extracted from the concentrates was under 80 per cent.; with present hand-washing the costs are from $8 to $9 per ton, and extraction about 90 per cent.; 95 per cent. could be got, but with large amounts of lime in the concentrates a 90 per cent. extraction is perhaps more economical than 95 per cent. would be. It was practically impossible with the Spence furnace to get uniform dead-roasts. After making futile attempts to remedy the defects of mechanical roasting the new management abandoned the system. Where 3 or 4 per cent., or perhaps 2 per cent., of sulphur can be left in, I imagine they will answer."

Writing again on September 30th, 1893, Mr. Hamilton Smith remarks:—" I think the chances are that no mechanical furnace will ever prove so successful at a mine for perfectly roasting sulphides as a reverberatory. The sulphides vary from week to week, so that hand-stirring in an open fire will always be more efficient. At large works where concentrates from various mines are mixed under the charge of a competent man, it is possible mechanical stirring may answer, but at one mine the owners cannot afford to pay very high salaries, consequently the man in charge of the furnaces and chlorination is generally (almost invariably) a man who works by rule of thumb. The most bothersome variation in the Alaska concentrates is the quantity of lime they contain. The more lime the larger the amount of salt required, and with considerable salt the volatilization of gold becomes notable, since we think the economical results are fully as good by increasing the salt with the contents of lime. In the old days chemists did not believe in a notable loss of gold by adding comparatively small amounts of salt in roasting, but experience clearly shows the old idea is wrong. The Frue vanners have probably been more successful at Alaska Treadwell than anywhere else, the tailings generally assaying less than 50 cents per ton leaving the vanners."

On page 189, Mr. T. A. Rickard, in a letter to the author, dated December 28th, 1893, states that the plant of the North Star mill, Nevada County, has been enlarged since the statistics from which the information given in Table I. with regard to it were compiled. It now includes a 6 feet wheel, 4 feet and 8 feet Pelton wheel, with no engine, 3 rock-breakers, 40 stamps, 12 Triumphs and 4 Frue concentrators.

On page 194, Table II. The South Clunes mill, Victoria, results are for 1890, the total milling cost is 2s. 3d. and not 2s. 11d., the difference being due to the cost for material, fuel, and repairs being 8½d. and not 1s. 4½d. as stated. The Saxon mill, Thames District, results are for 1892, they correspond with the statement from which they were taken, but ought to be transposed, being 1s. 2½d. for labour and 2s. 10½d. for supplies, the total of 4s. 1d. remaining unchanged.

On page 195, Table II. (continued). Hidden Treasure mill, Colorado, insert opposite water *nil*. Mr. Rickard gives the following working costs of the North Star mine in California for the years 1888, 1889, 1890:—52,708 tons were crushed, labour cost 1s. 3d., material and supplies 9d., water 1s. 3d., total 3s. 3d., or without water 2s. per ton.

On page 197, line 8, *for* "Table I." *read* "Table II."

On page 198, third line from bottom, *for* "Table I." *read* "Table II."

On page 199, the labour data to extend Table III. so as to include the North Star mill would be:—1 amalgamator, 12 hours at 16s.; 1 amalgamator, 12 hours at 12s. 6d.; 1 rock-breakerman, 10 hours at 10s. 6d.; 1 mechanic, 10 hours at 12s. 6d.; and 1 vanner attendant, 10 hours at 10s. 6d.

On page 200, line 26, *read* "£120; on the most unfavourable supposition we might fairly attribute," etc.

On page 202, the results of practice referred to in Gilpin Co. are taken from Mr. Rickard's article on "Variations in Milling."

On page 206, footnote, Messrs. McDermott and Duffield's *Gold Amalgamation*, page 5.

On page 218, line 2, *for* "Ingo," *read* "Inyo."

On page 220, line 24, *for* "Blaney," *read* "Blamey."

On pages 220 and 221, footnotes * and †, *for* "see pages 218 and 219," *read* "see page 220."

On page 257, line 6, *for* "10·37 ounces," *read* "10·34 ounces."

On page 257, line 7, should read "above assay value of 1·74 ounces of gold and 10·22 ounces of silver, representing," etc.

On page 257, line 13, should read "per ton amounting on 4·032 tons to 14s. 1d," etc.